DRAGLINE PERFORMANCE STUDY IN INDIAN COAL MINES

A THESIS SUBMITTED IN PARTIAL FULFILLMENT OF THE REQUIREMENTS FOR THE DEGREE OF

Bachelor of Technology
In
Mining Engineering

By

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Rourkela-769008

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CERTIFICATE

This is to certify that the thesis entitled **“DRAGLINE PERFORMANCE STUDY IN INDIAN COAL MINES”** submitted by **Sri Ankit Kumar** in partial fulfilment of the requirements for the award of Bachelor of Technology degree in Mining Engineering at the National Institute of Technology, Rourkela (Deemed University) is an authentic work carried out by him under my supervision and guidance.

To the best of my knowledge, the matter embodied in the thesis has not been submitted to any other University/Institute for the award of any Degree or Diploma.

Date:

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I will be failing in my duties if I didn't mention reference to and inspiration from the works of others whose details are mentioned in reference section. I acknowledge my indebtedness to all of them.

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Date: **ANKIT KUMAR**

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ABSTRACT

Draglines have been abundantly used in coal mining for decades, either as stripper or stripper and coal extractor. Since this equipment has certain inborn advantages, which their challengers do not, they must be operated in a 24-hour day and night manner for high productivity and low costs. In India, the development of giant surface mining ventures like Bina and Jayant removes large volume of overburden in shortest possible time to achieve higher coal production targets (upto 10 million tonnes per annum). This has led to major changes in overburden/ interburden excavation technology in surface coal mines from shovel mining to that of draglines. Coal India Limited (CIL) has now standardized the draglines in two sizes, which are 10/70 and 24/96 for their mines. Most of the mines depend 24 hours a day, 7 days a week on the dragline working. In many coal mines, the mine's output is totally dependent on the dragline's performance since it is the only primary stripping tool. For these reasons, design of draglines require importance placed on developing component's with high levels of reliability and predictability so that repairs and replacement of components can be scheduled at times that will least affect the overall mining operation. Before deploying draglines in mines, various factors have to be considered for selection of suitable size of the dragline and other parameters. Different parameters are used to determine the production and productivity of draglines. Also availability and utilization of different draglines have been calculated and their performance was studied and various reasons of decreasing performance are pointed out. In this thesis these points are discussed in detail.

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Chapter 01

INTRODUCTION

1.1. Dragline mining

There is a continuous increasing need for energy and to fulfill this demand, coal is required in huge quantity. Mother earth has been kind enough to homogeneously spread this raw material throughout its crust. A considerable portion of coal is produced by adopting surface mining methods. To meet the demands of thermal, cement and other industry sectors the trend in production of coal has been increasing. In order to have high production with minimum cost, Draglines are being used in coal mining. This equipment has been chosen because of its certain advantages like economically excavate deposits at greater and greater depths. Dragline is a one of the surface mining equipment which is used to excavate materials and is so designed so that it can excavate below the level of the machine also. Dragline working can be divided into two parts: Digging and Walking. Among them walking is a steady process on which the mine design team has little control. Almost all walking draglines take a step of approximately 2 m within a time period of 0.75-1 min. The design of strip panels, equipping a specific unit with one operator's room on the desired side or with two on both sides and the management's strategy in coal loading operation largely affect the frequency and the length of long deadheading periods, during which the unit is unproductive (Erdem et al., 2003).

Dragline used in surface mining is heavy equipment. They are mostly built on-site for strip mining operations to remove overburden and coal and they are largest mobile land machine ever built. It consists of large bucket which is suspended from a boom with wire ropes. The bucket is maneuvered by means of ropes and chains. The hoist rope is powered by electric motors and drag rope is used to draw bucket assembly horizontally.

Various operations of the bucket for the desired purposes are controlled by maneuvering of the hoist and drag ropes. The dump operation of the dragline consists of positioning the bucket above the material to be excavated and then lowering it so as to drag it on the surface of the material. The bucket is then lifted by the hoist rope and a swing operation is then performed to move the bucket to the place where the material is to be dumped, drag rope is then released causing the bucket to tilt and empty.

A typical dragline has volume ranging from 30 to 60 cubic meters and length of the boom ranges from 45 to 100 meters. Their power consumption is so great that they have direct connection to high voltage grid ranging from 6.6 to 22 kv. The movement of the dragline is accomplished by walking, using feet. Maximum speed is only few meters per minute. For larger distances disassembly is generally required. Mining draglines are not moved frequently as their reach can work a large area from one position.

A dragline is most efficient for excavating material below the level of their base. It can dig above itself also but inefficiently. It is not suitable to load piled up material as a shovel.

Draglines are popular with many mines due to their reliability and extreme low waste removal cost. However, they have very limited application due to their boom height, boom length and dig-depth.

Nowadays keeping in view of the demand of increasing productivity draglines having longer booms and large bucket capacity are being selected. However to obtain low cost per cubic meter it is imperative to operate the dragline in a scientific manner for which constant supervision, good overburden preparation, and preventive maintenance of dragline and selective proper bench height of overburden is very much required. In order to achieve higher coal production targets, it is necessary to remove large volumes of overburden in shortest possible time and therefore it has been found that Draglines are the best suited equipment to perform this type of work.

1.2. Dragline mining in India

A dragline is a massive earthmoving machine. It is predominantly used in open-cast coal mines to strip the overburden covering the coal but is also found on a much smaller scale in civil operation. The larger draglines are used in strip mining to remove the soil layers covering coal. The smaller machines may be used to de-silt canals and dams. Draglines have gained popularity in India for overburden stripping because of their flexibility and high production rate. Since many giant projects are coming up more in the coal sector in recent times, the shovel-truck mining faces big challenges in fulfilling the production demand. Hence the Indian surface coal mining has been switching over from shovel-truck mining to dragline mining for removal of overburden/ interburden in most of large sized coal mines to accommodate high rate of overburden removal and subsequently, high production rate with low cost of production. Dragline based stripping systems bring an economical saving up to 40 percent compared to shovel-truck method (Gol basi, 2012). These have been the first choice for excavation under favorable geo-mining conditions because of the huge capacity to handle material at a very low cost.

Draglines are used where the burden that is excavated, must be transported over a short distance only i.e. maximum approximately 100m. Thus, it is ideal for strip mining application where the width of strip i.e. exposed coal rarely exceed 50-60m (Hartman, 1992).

In the coal mines of India Draglines were initially introduced in early 1960's. The first walking dragline was commissioned in 1961 at Kurasia Open cast of erstwhile M/ s National Coal Development Corporation (NCDC). It was page model 734E with 11.47 cubic meter bucket capacity. Two more machines 6/45 P&H model 1855 were commissioned at Kurasia and south Balanda. Bigger machines of 30 cubic meter bucket capacity Marion model 7800 were commissioned at Bisrampur opencast mine of NCDC in 1964 and 1967 respectively. Earlier draglines were of smaller capacity and smaller reach except the Marion 7800 at Bisrampur. Mostly 5/45, 10/70 and 15-20/90 were deployed. Later on 20/90 and 24/96 draglines were introduced in the mines of Moher basin in Singrauli CF in the mines of CCL, which formed the present NCL.

Out of 40 draglines presently working in the country majority (38) is in Coal India Ltd. Whereas 2 are in Singareni. Few are under order or erection. Two numbers of 62/100 draglines are being erected at Moher mine of M/ s Sasan (Reliance), which has the distinction of having deployed largest of these machines.

Coal India Ltd. has now standardized the draglines in two sizes, which are 10/70, and 24/96 for their mines. The economic life of a dragline has been assumed by CIL to be 25-30 years.

Northern Coalfields Limited (NCL) is the only subsidiary company of CIL, where the entire coal production is mined by opencast mining method. Another unique feature of the company is that about 40 per cent of the large volume of excavation is done with the help of larger walking draglines. Draglines are used in all the mines of NCL except in Jhingurda, Kakri and Gorbi (Vemana, 2012).

1.3. Aim of the Paper

The aim of this paper is to measure and calculate the projected output, calculation of ownership, operating cost and cost per tonne of coal exposed by dragline by the combination of various parameters collected during field study and acquired from other sources and to study their performance on the basis of draglines availability and utilization. The secondary goal is to Develop a computer based program (in MATLAB) on dragline bench designing (keeping the cut-width constant) and determine whether or not rehandling is required and projection of annual production of overburden, calculation of ownership, operating cost and cost per cu.m overburden handle of the dragline.

1.4. Objectives

1. Literature review on:

- ❖ Draglines
- ❖ System of working of dragline and methods of working of dragline
- ❖ Draglines used in India

2. Dragline balancing diagram and developing a computer based program (in MATLAB) on dragline bench designing (keeping the cut-width constant) and determine whether or not rehandling is required.

3. To find total time of availability & utilization of dragline

4. Projection of annual output, calculation of ownership, operating cost and cost per tonne of coal exposed by dragline.

5. Develop a computer based program (in MATLAB) on projection of annual production of overburden, calculation of ownership, operating cost operating and cost per cu.m overburden handle of the dragline.

1.5. Methodology

The specific objectives were achieved by the adoption of following methods:

1. Critical review of available literature

2. Visit to dragline mines for collecting and recording various parameters required for the study

3. Thorough calculation and computer programming to achieve the goals and objectives

4. Attend to various seminar on dragline

Chapter 02

LITERATURE REVIEW

2.1. History and present

The dragline was invented in 1904 by John W. Page of Page Schnabel Contracting for the purpose of digging the Chicago Canal. In 1912 it became the Page Engineering Company, and a walking mechanism was developed a few years later, providing draglines with mobility. Page also invented the arched dragline bucket; a design still commonly used today by draglines from many other manufacturers, and an archless bucket design in 1960s.

In 1910 Bucyrus International purchased the manufacturing rights for the Heyworth-Newman dragline excavator and with this he entered into the dragline market. Their "Class 14" dragline was introduced in 1911 as the first crawler mounted dragline. In 1912 Bucyrus helped pioneer the use of electricity as a power source for large stripping shovels and draglines used in mining.

In 1914 Harnischfeger Corporation, (established as P&H Mining in 1884 by Alonzo Pawling and Henry Harnischfeger), introduced the world's first gasoline engine-powered dragline. An Italian company, Fiorentini, produced dragline excavators from 1919 licensed by Bucyrus.

In 1939 the Marion Steam Shovel Dredge Company (established in 1880) built its first walking dragline. The company changed its name to the Marion Power Shovel Company in 1946 and was acquired by Bucyrus in 1997. In 1988 Page was acquired by the Harnischfeger Corp., makers of the P&H line of shovels, draglines, and cranes.

Today, draglines are extensively used in strip mining of coal throughout the world. However, it has found wide range use in non-coal sector also, which includes surface mining of bauxite, phosphor, oil shale and tax sands. In the USSR, draglines are deployed widely for rehandling and sticking of 0/B spoil dumped by rail transport system. Occasionally, but rarely, these machines are used for loading into dumpers or bunkers as well for which special arch less buckets are available. In underwater digging such as for collecting sand and gravel, draglines are quite equipped with perforated buckets.

Presently there are five major manufacturers of draglines. They are Bucyrus Erie (US), Page (US), Marion (US), Rapier and Ransom (UK) and the Soviets. In India, Heavy Engineering Corporation is progressively manufacturing W-2000 model walking dragline indigenously in collaboration with Rapier and Ransom. Draglines used in open-cast mining typically range in size from machines equipped with 5 cubic meter drag buckets on 35 meter booms to the Bucyrus – Erie model 4250W, which is equipped with a 168 cubic meter drag-bucket on a 94.5 m boom. The longest boom length (121.9 m) dragline is offered by Bucyrus Erie, page, as well as, Marion. The largest boom from Ransom and Rapier is 105.5 m. The Soviets commissioned a long boom dragline with 120 m length during 1989. Works are now in progress to construct draglines having bucket capacity is high as 200 cubic meters. The current trend is to have machines with high bucket capacity and with short boom length. Apart from enhancing productivity and flexibility this arrangement can, most certainly, lend a degree of safety to the overall working conditions.

Most mines depend on the dragline 24 hours a day, 7 days a week. In many coal mines, it is the only primary stripping tool and the mine's output is totally dependent on the dragline’s performance. For these reasons, dragline design requires emphasis placed on developing component’s with high levels of reliability and predictability so that repairs and replacement of components can be scheduled at times that will least affect the overall mining operation.

Another critical designed consideration is that most repairs must be performed away from shop facilities. Although the dragline is a mobile piece of equipment, its enormous size prevents bringing to the shops for maintenance and repairs as s common with trucks and o her mine equipment. The designer must ensure that components are really accessible and that portable tools and rigging equipment are available for any eventuality.

2.2. Typical specification:

1. Dragline used in open cast mining has size ranges from 5 cubic meter drag bucket on a 35 meter boom to the Bucyrus- Erie model 4250 w, which is equipped with a 168 cubic meter drag bucket on a 94.5 meter boom.

2. The longest boom length of 121.9 meter boom length dragline is offered by Bucyrus, Page, as well as, Marion.

3. The largest boom of Ranson, and Rapier is 105.5 meter.

4. The soviet commissioned a long boom dragline with 120 meter length during 1989.

5. There are many works in progress nowadays to construct draglines having bucket capacity as high as 200 cubic meters.

6. The current trend is to have machine with high bucket capacity and with short boom length that increases degree of safety to the overall working condition apart from enhancing productivity and flexibility.

2.3. The Big Muskie

Big Muskie (Model: 4250-W) was the largest dragline manufactured by Bucyrus- Erie in 1969. It was the world's largest mobile earth-moving machine. The other machines which are close to the Big Muskie are "The Captain Marion 6360" a stripping shovel and the German "The Bagger 288" a bucket wheel excavator. Big Muskie was largest single bucket digging machine ever created.

It was owned by the Central Ohio Coal Company mining brown coal. It weighted 13000 tons and standing nearly 22 stories tall (67.82 m). Its bucket capacity was 170 cubic meters and boom length 94 meter. It had hydraulically driven walker feet and travelled very slowly at the speed of 4.5cm/sec, operated at 13800 volts. Big Muskie became unprofitable to operate because of high sulphur content in brown coal, increased electricity cost, hostile public opinion against strip mining and due to enormous environmental costs. No other coal company was prepared to buy it due to massive cost involved into dismantling, transporting and reassembling the machine. It became inoperative in 1991 and it remained idle for eight years. Many historians wanted to convert the machine into a museum but in late 1999 it was broken down on the site and scrap were sold for $700,000. The massive bucket was preserved and relocated to newly constructed Miners Memorial Park in Morgan County.

2.4. Conditions for operation of dragline

1. Gradients should be flatter than 1 in 6

2. Seams should be free of faults & other geological disturbances

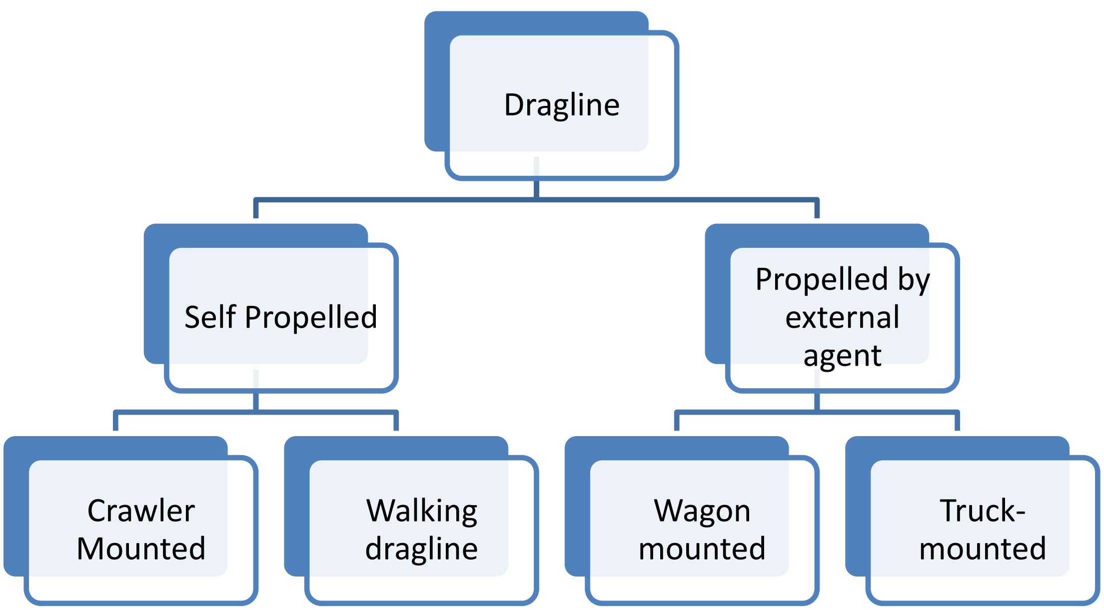
3. Adequate Strike length of the quarry, should not be less than 1 km

4. Thick seams with more than 25m thick are not suitable

5. A hilly property is not suitable

6. Overburden can easily be excavated after blasting

2.5. Classification of draglines



2.6. System of working

In a typical cycle of excavation, the bucket is positioned above the material to be excavated. The bucket is then lowered and the dragrope is then drawn so that the bucket is dragged along the surface of the material. The bucket is then lifted by using the hoist rope. A swing operation is then performed to move the bucket to the place where the material is to be dumped. The dragrope is then released causing the bucket to tilt and empty. This is called a dump operation.

The bucket can also be 'thrown' by winding up to the jib and then releasing a clutch on the drag cable. This would then swing the bucket like a pendulum. Once the bucket had passed the vertical, the hoist cable would be released thus throwing the bucket. On smaller draglines, a skilled operator could make the bucket land about one-half the length of the jib further away than if it had just been dropped. On larger draglines, only a few extra meters may be reached.

Draglines have different cutting sequences. The first is the side cast method using offset benches; this involves throwing the overburden sideways onto blasted material to make a bench. The second is a key pass. This pass cuts a key at the toe of the new highwall and also shifts the bench further towards the low-wall. This may also require a chop pass if the wall is blocky. A chop pass involves the bucket being dropped down onto an angled highwall to scale the surface. The next sequence is the slowest operation, the blocks pass. However, this pass moves most of the material. It involves using the key to access to bottom of the material to lift it up to spoil or to an elevated bench level. The final cut if required is a pull back, pulling material back further to the low-wall side.

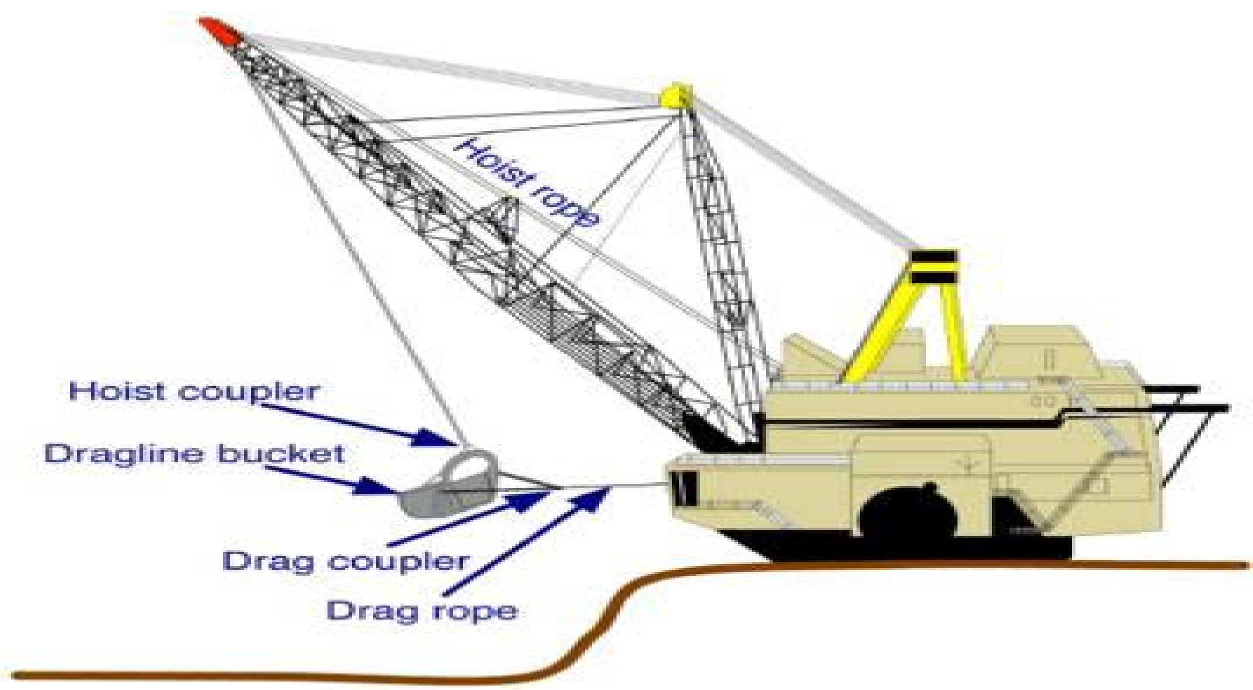


Fig. 2.1: Line diagram of dragline

The operating cycle of the dragline consists of five basic steps:

1. The empty bucket is positioned, ready to be filled.

2. The bucket is dragged toward the dragline to fill it.

3. The filled bucket is simultaneously hoisted and swung over to the spoil pile. If the swing motion must be slowed to permit hoisting, the dragline is said to be hoist critical. When hoisting to the dump position is completed before the boom is in position to dump, the dragline is said to be swing critical.

4. The material is dumped on the spoil.

5. The bucket is swung back to the cut while simultaneously being lowered and retrieved to the digging position.

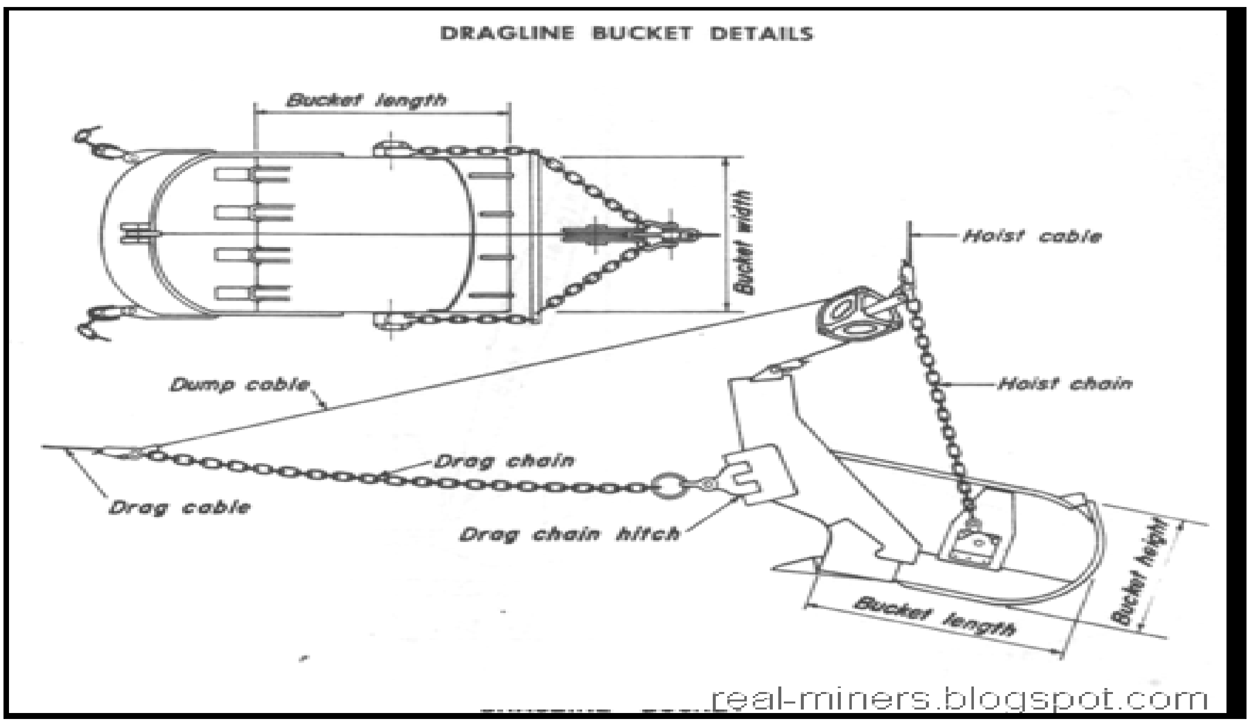


Fig 2.2: Parts of dragline bucket

2.7. Dragline stripping methods

The stripping cycle of dragline begins with the dragline cutting a trench, referred to as the key cut, along the newly formed highwall. The distance from the previous key cut position to the new position is referred to as the digout length. The key cut is made to maintain the panel width and uniform highwall. Without a key cut, the panel width would narrow with each subsequent digout, because the dragline could not control the bucket digging against an open face. The dragline deposits the key cut material in the bottom of the mined-out pit off the coal and against the previous spoil pile. More stable spoil from the key cut may be placed in the very bottom next to previous spoil to form the buckwall which provides a more stable spoil slope that can be steepened if deemed necessary.

When the key cut has been completed, the dragline is moved to new position to complete excavation of the digout. This is known as the production cut, and the material is cast on top of the key cut spoil. When the digout has been completed, the dragline is moved to next position, the beginning of the next stripping cycle (next digout).

Efficient dragline operation is realized by minimizing the time required to position, drag, and dump while synchronizing the swing and hoisting motions. Synchronization of hoisting and swinging is dependent on the time the boom is in motion.

Full Key Cut vs. Layer Cut: At the beginning of a new digout, the dragline generally is placed directly over the toe of the new highwall to be formed. From this position, the dragline can establish a uniform and safe highwall if the burden is sufficiently stable.

In this position, the dragline excavates the key cut which is more than the width of the bucket at the bottom of the cut. When the cut has been completed, the dragline moves over to make the production cut. The two positions generally are required because of the limited reach of the dragline in relation to the panel width being stripped. Large draglines, operating under ideal conditions, may be able to excavate the total digout from the one position over the highwall. Such situations are the exception, not the rule.

When operating conditions permit excavation of the dig out from one position over the highwall, the dragline generally excavates the digout in layers. The key cut is formed, one layer at a time, by excavating along the highwall before the completion of each layer. Cutting in layers can be performed from the production cut position; however, the high wall slope will require dressing by dozer while the dragline is digging. Under such circumstances, some mines also have adequately dressed the highwall by dangling a heavy section of chain from the bucket and dragging the chain along the wall. Other mines, because spoil area is critical, have progressively stepped the dragline toward the spoil while excavating in the layer cut method. This procedure has the tendency to pack spoil as tightly as possible on the spoil slope.

Layer cutting generally increases dragline productivity with a corresponding decrease in operating cost. Increased productivity is realized by progressively decreasing the average swing angle as the dragline walks in the direction of the spoil pile.

Dragline Panel Width: Panel width is defined as the width of the cut taken by the dragline, as it progresses from digout to digout, along the highwall from one end of the pit to the other. Panel width, one of the most important parameters affecting dragline productivity, is influenced by depth of overburden, dragline boom length, hoist and swing time, and available spoil area. Since panel width becomes the available operating area in the pit bottom, coal loading operations are also affected.

Several operational factors must be considered in the selection of panel width. A wide pit generally is favorable for coal load-out and permits greater safety for men and equipment. The minimum practical pit width is dictated by the maneuverability of coal loading and hauling equipment.

If the available space for placement of spoil is critical, such as might occur when crowding spoil to open haul roads through the spoil, narrow panels permit greater flexibility to deal with such problems. In general, the wider the panel, the less dragline walking time is required.

Productivity variations, because of panel width, are directly related to whether or not the dragline is swing critical. Small draglines can become swing critical at panel widths less than the width required for practical coal operations in the pit. Their cycle time also increases dramatically. Larger draglines may not become swing critical until the panel width exceeds 50 m (150 ft).

Bench Height: The height above the coal seam at which the dragline is positioned is defined as the bench height. Selection of the bench height is based on numerous operational factors and topographic restraints.

The complex relationship of bench height (which could be equal to overburden depth), panel width, dragline dumping reach and dumping height, as well as material characteristics such as swell and angle of repose, influence greatly the dragline's capability to dispose of burden off the coal. The dragline's digging depth, while related to burden depth, rarely becomes a factor in dragline performance.

The bench height must be selected primarily on the basis of fitting the dragline's specific characteristics to the required pit geometry. In general, the bench height should be as high as possible within the limit of required dragline reach.

Undulating topography may complicate a simplified selection of bench height. Two alternatives are available to alleviate the problem:

1. The dragline can be used to cut and fill to develop a common bench elevation. Cutting termed chopping or overhand digging, increases the cycle time and reduces the bucket fill factor, thereby reducing effective productivity. Fill material must be rehandled, thus reducing overall production. Chopping has very special advantages: the dragline reach required may be shortened, rehandling of burden may be avoided, fill may not be required to create a level working surface, a level return path for deadheading can be provided, and subsoil can be placed back in its relative position on top of the spoil.

2. Auxiliary equipment can be used to perform the cut and fill operation. Care must be exercised to ensure that filled areas are stable. Utilizing auxiliary equipment offers the benefit of freeing the dragline for its primary function of stripping burden from the coal.

Whether the dragline cuts and fills its working pad or auxiliary equipment is utilized to prepare a level working surface depending on several variables. Dragline chopping decreases overburden stripping productivity and may involve abnormal wear and tear on dragline equipment. Auxiliary equipment for prestripping adds to the capital and operating costs of operations. Depending on the thickness of the material to be chopped, the cost differential between chopping and using auxiliary equipment is not likely to be high when small draglines are compared. For large draglines, prestripping with auxiliary equipment will very likely be preferable.

Digout Length: The selection of digout length, the length between major digging cycles, is based on the relationship of the dragline's operating characteristics with respect to pit geometry. In general, the digout should be as long as possible. However, dragline size may greatly influence digout length for specific pit geometry. For example, digout length is sensitive when using a dragline with slow hoist speed working in deep overburden. Spoil critical pits may utilize a less than desirable digout length in order to pack the maximum material onto the spoil bank.

In general, a long digout with respect to dragline size reduces cycle time and increases productivity because more material is loaded under the outer end of the boom than near the fairleads of the dragline. A good dragline operator will try to fill the bucket within two and a half to three times the bucket length. Cycle components of retrieving for bucket loading, bucket dragging, payout for dumping, and swing angle all are decreased as the digout length increases. Longer digout lengths also reduce the nonproductive time required for repositioning the dragline on the succeeding digout. Obviously, digout length should not be so long as to require the dragline operator to cast the bucket beyond the limit of the boom.

Walking Patterns: In a dragline operation, two separate walking cycles are involved: deadheading and walking within the digout. When a panel has been completed, there are two options available for the dragline. One, the dragline can wait for the coal to be mined to the end of the pit, then turn around and begin the next panel in the opposite direction. This procedure is termed laying over at the end of a panel. Two, the dragline can turn around and travel part or all of the way down the panel to begin the next cut. This procedure is termed deadheading. If the dragline travels part way down the panel, it cuts into the next panel and strips in the opposite direction. If the dragline travels to the other end of the pit, it cuts in to the next panel and strips in the same direction.

Whether a dragline lies over or deadheads depends primarily on the production time lost. Contractual requirements, such as exposed coal inventory, may eliminate the layover option and force deadheading. Some mines opt to lay over at the end of a panel because ground conditions are not favorable for deadheading. Other mines limit layover to a maximum of two shifts. If waiting for coal production involves two or more lost dragline shifts, the dragline will be deadheaded.

When deadheading is feasible, the decision should be made on the basis of minimum lost dragline production. Deadhead time is based on 33% of the specified dragline walking speed. Such a large discount factor must be used to account for various delays in deadheading, such as maneuvering, cable handling, ground preparation, and minor breakdowns.

The greater the digout length, the less walking time will be required per panel. Repositioning in the digout can affect cycle time. Therefore, walking patterns must be considered when selecting digout length. Time spent in repositioning the dragline can be estimated by discounting the walking speed of the dragline. Generally the discount factor is much less than the factor utilized for deadhead estimates. Based on observations by the author, a discount to 15 to 20% is appropriate.

Pit Shape: The new dragline pit begins with the initial cut, termed the box cut, made along the outcrop, sub crop, or property boundary. To open the box cut, excavated material is spoiled to one or both sides of the cut. The material lying on the newly created highwall must be moved or spread out evenly by auxiliary equipment. The material lying on the cut wall that will become the spoil side may, or may not, have to be moved depending on reclamation requirements.

Because of rolling topography, the mine engineer may be inclined to design the box cut along a uniform contour. Generally, succeeding cuts are designed parallel to the box cut. As a result, this type of pit develops a meandering design. Obviously, outside curves provide more spoil area. Depending on depth of overburden, panel width, radius of curvature, and operating parameters, severe operating problems may occur on inside curves. Dragline cycle time will increase, spoil crowding will occur, and coal may be lost by being covered with spoil.

To remedy the problems caused by inside curves, several options can be considered. Panel width may be decreased, material may be cast short and rehandled by extending the bench, a small auxiliary dragline may be utilized on the spoil to pull back excess spoil, the spoil pile may be steepened, or the pit may be straightened by stripping a series of short panels. Generally, the most favorable solution is to straighten the pit. Spoil steepening is also an effective method for disposing of relatively small amounts of excess spoil. The dragline bucket is positioned on the spoil slope where steepening is desired and dragged down and across the top of coal. The bottom part of the spoil pile is steepened and coal is cleaned in the process. Digging efficiency during this process is reduced, cycle time increased, and rehandling reduces effective productivity. Steepened spoil slopes may present special hazards to equipment and personnel because they are more prone to failure.

Spoil Patterns: There are three basic methods of spoiling. When using short digouts and casting at near 90^0 angles, a uniform ridge line can be created. This configuration makes maximum use of the available spoil room. As the digout length is increased, uniformity of the ridge line is lost and individual peaks of spoil are created.

With sufficient spoiling area, the dragline operator may cast material from both the key cut and production positions at angles less than 90^0. While the dragline stripping cycle will improve, spoil piles appear to be ragged and irregular. An aerial view of the operation will show a definitive pattern to the irregularity. In reality, spoil peak grading will be reduced by this method of spoiling.

Dragline cycle time can be reduced by dumping the loaded bucket on the fly, that is, before the dragline swings to the ultimate dumping position. This procedure, termed radial casting, gives the spoil a cross bedded appearance. Provided that there is sufficient spoil room, radial casting tends to spread the spoil more effectively, reducing spoil grading costs.

Since distance between spoil ridges is equivalent to the panel width, narrower panels will reduce spoil grading costs. However, such reductions in cost generally will be offset by increases in dragline operating cost if the dragline is not swinging critical.

2.7.1. Simple sidecasting method

This is the simplest form of strip mining, which involves excavation of the overburden in a series of parallel strips. The strips are worked in a series of blocks. The 0/B from each strip is dumped into the void left by the previous strip after the coal mineral has been mined. It is customary to start the excavation of each block by digging a wedge shaped key cut with the dragline standing in line with the new high wall. From this position, the machine can most easily dig a neat and competent high wall. The nearest high wall is affected by starting the out with the dragline in line with the crest and moving it as the out gets deeper, ending with the machine in line with the toe of the new high wall. By this means, the slope angle of the new high wall can be closely con rolled. The width of each strip is usually designed so that the material from the key cut can be thrown into the previous cut without the need for rehandle.

When the key cut has been completed, the dragline is moved close to the old high wall edge from where it can excavated the reminder of the blocks. With a suitable selection of bench height and block width, as well as, proper reach, casting can be done dear off the coal bench.

However, more often than not, the spoil pile touches the crest of the coal seam for obvious advantages mentioned early. Associated demerits are also present. Rehandling is no intended as it jeopardizes the economy of operations. Advance benching with this method is also practiced due o reasons already mentioned.

The manner in which a dragline must be applied to dispose of the material is of greater significance in affecting dragline productivity. In the simple case shown in the Fig. the dragline sets on the top of the material to be excavated and swings through an arc of between 45 to 90 degrees to dump the material. A typical average cycle time for the operation is 45 seconds.

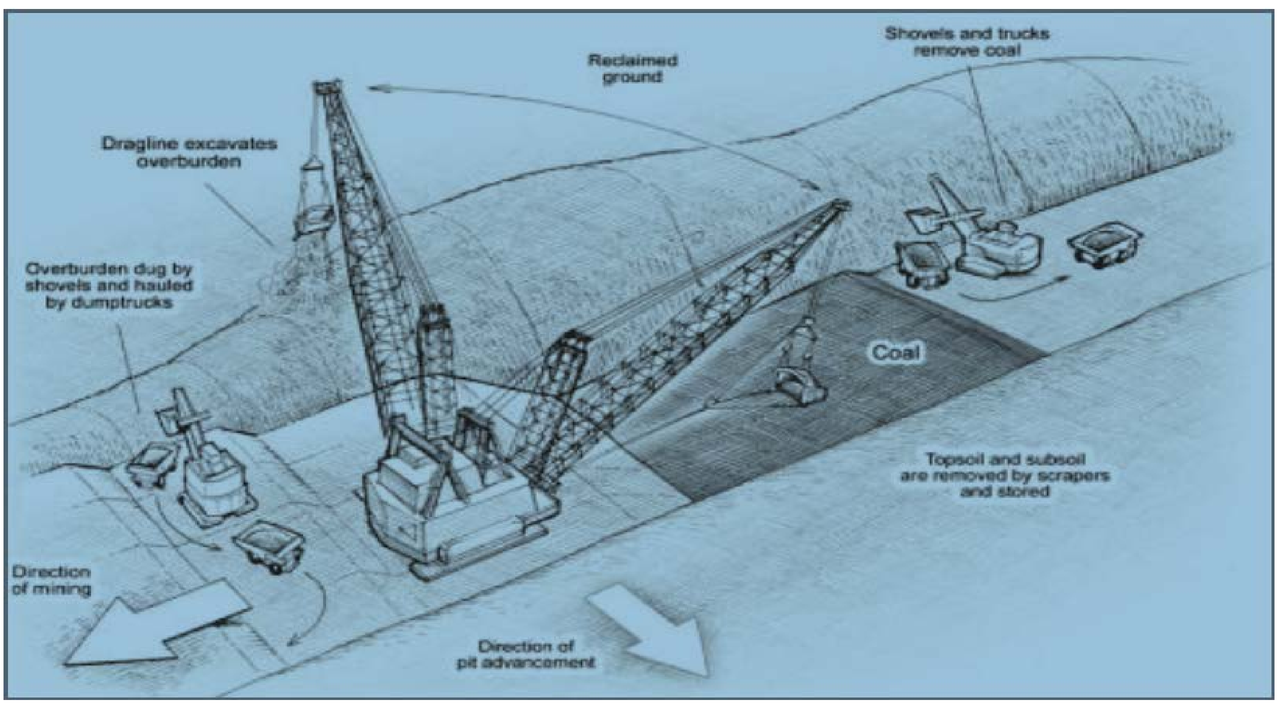


Fig 2.3: Simple sidecasting method

To obtain maximum reach, it is necessary to work the machine as close as possible to the high wall crest. In addition to the obvious risks to very expensive equipment, this practice reduces the degree of blasting which can be employed. In order to preserve a satisfactory edge from which to work, several mines 'buffer shoot' two or three strips ahead of the dragline. Buffer shooting is undoubtedly less efficient than shooting to a free face and no advantage can be taken of the material cast by the shot.

2.7.2. Dragline Extended bench method

Where overburden depth or the panel width exceeds the limit at which the dragline can sidecast the burden from the coal, a bridge of burden can be formed between the bank and the spoil which effectively extends the reach of the dragline. The bridge extends the bench on which the dragline is operating. The bridge is formed by material falling down the spoil bank or by direct placement with the dragline. To remove the bridge material from the top of coal, it must be rehandled.

Extended bench systems are adaptable to many configurations of pit geometry. In this method the dragline forms its working bench by chopping material from above the bench and forming the bridge, then moving onto the bridge to remove it from top of coal. The primary dragline strips overburden and spoils it into the previously excavated panel. This material is leveled, either by tractor-dozers or the secondary dragline, to form the bench for the secondary dragline. The secondary dragline first strips material near the highwall, then moves on to the bridge to move the rehandle material. In a two-dragline system, one machine must operate at the pace set by the other. Therefore, mine design must consider their respective capacities when assigning respective digging depths. The primary dragline strips overburden to the top of the first seam. Coal is removed, then a small parting dozed into the pit and the second coal seam removed. The secondary dragline strips the large interburden to the third and final seam. Extended bench systems must be designed carefully in order to maximize the dragline(s) productivity and to minimize the amount of rehandle.

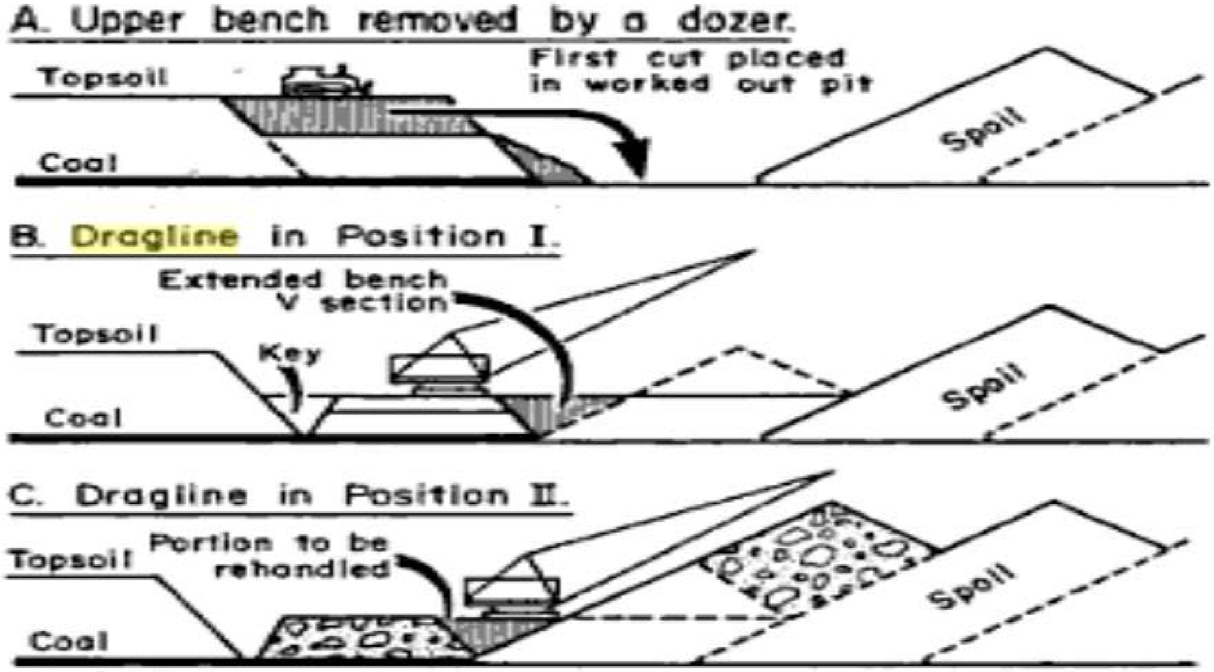


Fig 2.4: Positions in extended bench method

2.7.3. Dragline Pull-Back Method

Occasionally, overburden to be stripped will be beyond the capacity of the dragline to spoil off the coal by any of the previous methods described. In this case, a secondary dragline can be placed on the spoil bank to pull back sufficient spoil to make room for complete removal of overburden.

Generally, rehandle volume is greater for the pull-back than an extended bench method of operation. However, it may also serve to level spoil piles in addition to providing more spoil area for the primary dragline. If the overburden/interburden is generally beyond the capability of draglines working on the highwall, the pullback method would seem to be a solution. However, great care must be given to the design of this method because of the inherent hazards of operations. Spoil slopes can be unstable, more so during periods of severe rainfall.

Draglines frequently are utilized to strip overburden from deeper coal seams than originally intended. Occasionally, spoil slopes cannot be maintained at designed angles. Various methods have evolved to stack more material into the spoil bank to alleviate these problems. The more common methods are described briefly:

1. Buck walls involve building the base of the spoil adjacent to the pit with competent material so that a steeper spoil slope near the base can be maintained.

2. Coal fenders require leaving a small wedge of coal untouched in the pit so that more spoil can be packed on the spoil slope.

3. Outside pit involves modifying the pit shape in order to develop the outside curve concept which increases the spoil area relative to the stripping area.

2.7.4. Tandem machine systems

Tandem machine stripping of two coal seams and/or deeper overburden can be done by using a dragline with a second system which can be another dragline, shovel, scraper dozers, etc. This system has two advantages as:

(i) One machine removing both overburden (thickness 100ft) and interburden (thickness 20ft) will probably be less efficient than two machines – one designed specifically for overburden removal and the other for interburden removal.

(ii) Production capacity, provided the operations were well planned, in tandem machine operations may be greater than that in single machine operations under similar operating conditions.

Disadvantages of the tandem system include when loading machine is down, the trailing machine will often be idled. Such situations can however be minimized by good planning.

2.8. Drilling and Blasting of Overburden for Draglines

Overburden drilling and blasting is more critical for dragline stripping than for shovel digging. Shovels have the ability to crowd the dipper into the bank, providing leverage to dig difficult or poorly blasted material. Draglines have leverage only by dragging the bucket over the material. Such leverage is translated to severe strain on the bucket lip and teeth. In poorly blasted material, dragline productivity can drop more rapidly than that of shovels working in similar material.

Selective placement of explosives and blasting agents may be critical to the surface coal mine operation. Many coal seams are overlain with sedimentary beds of varying hardness and thickness. Improper placement of the charge in the blasthole can cause blast energy to travel along planes of greater weakness and through softer material. Under such conditions, harder beds of material will tend to break in large blocks or fragments. To ensure adequate placement of the blast charge, it is necessary that drill operators log differences in material or drill penetration rates and provide this information to the blasting foreman.

For dragline stripping, there are two general methods of blasting overburden in common use. One method utilizes a blast-hole pattern with a buffer zone to contain the blasted material against the highwall. The advantage of this pattern is to contain the blasted material within the dragline working area and avoid large broken material that must be handled with difficulty. The widths of the buffer zone, combined with the powder factor, are critical elements in efficient utilization of this method. This method is useful, especially if the dragline is performing a chop cut prior to the key and production cuts.

The other method of blasting overburden is similar to the standard open pit blasting procedure. Its purpose is to blast as much material into the spoil area as possible, thereby reducing the amount that must be stripped by the dragline. The resultant advantage is debatable in the author's opinion, since considerable grading is necessary before the dragline can begin casting. Frequently the dragline is called upon to rebuild its working pad by retrieving material from the pit. Time lost in pad preparation may completely offset the original reduction in stripping volume. If the dragline can safely work on the spoil side of the pit, building its working pad ahead on the spoil, there may be justification for blasting material from highwall to the spoil. Increased costs of explosives, pad building costs, and highwall scaling delays must be weighed against the difference in overburden volume to be stripped (SME handbook….).

2.9. Draglines in use in India

2.9.1 Draglines in CIL as on 1.4.2013:

Table 2.1- Eastern Coalfields Ltd. (ECL)

	Project	Capacity	No. of D/L	CIL Plant No.	Date Of Commissioning	Cumulative worked Hrs. till 31.03.2013	Geo-mining conditions
1.	Sonepur Bazari	24/96	1	EXC 1885	09.11.1996	109153	Multi seam deposit, bottom medium thick seam exposed by dragline
Total for ECL			1				

Table 2.2- Western Coalfields Ltd. (WCL)

	Project	Capacity	No. of D/L	CIL Plant No.	Date Of Commissioning	Cumulative worked Hrs. till 31.03.2013	Geo-mining conditions
1.	Sasti	20/90	1	EXC 2214	17.06.1992	129428	Single thick seam (16-22 m)
2.	Umrer	6.9 CuM	1	EXC 074	13.08.1965	246729	Multi seam deposit, bottom seam is thickest. Shovel-dumper for upper seams. 22 Small dragline used for rehandling
3.	Ghugus	24/96	1	EXC 1610	08.12.1991	119129	Single thick seam (16 to 22 m) developed in two sections
Total for WCL			3				

Table 2.3- Bharat Coking Coal Ltd. (BCCL)

	Project	Capacity	No. of D/L	CIL Plant No.	Date Of Commissioning	Cumulative worked Hrs. till 31.03.2013	Geo-mining conditions
1.	Block II	24/96	1	EXC 1575	04.12.1988	90082	Mid seam of coking coal worked. OB dumped in coal bearing area to be removed later
2.	South Tisra	6.5/45	1	EXC 1686	08.12.1989	88251	--------------------
Total for BCCL			2				

Table 2.4- Mahanadi Coalfields Ltd. (MCL)

	Project	Capacity	No. of D/L	CIL Plant No.	Date Of Commissioning	Cumulative worked Hrs. till 31.03.2013	Geo-mining conditions
1.	Lajkura	10/70 10/60	2	EXC 1571 EXC 89	26.02.1990 12.01.1970	123836 175714	OB above a thick seam interbanded seam taken by dragline
2.	Samleshwari	10/90	1	EXC 1984	01.07.1992	117841	---------------
3.	Belpahar	10/70	1	EXC 1580	10.10.1990	135491	Parting between two seams taken by dragline
4.	Bharatpur	20/90	1	EXC 1280	01.05.1989	115784	A thick seam (10-16 m) is split into 3 to 4 splits in part of the area. Mostly, single seam working
Total for MCL			5				

Table 2.5- South Eastern Coalfields Ltd. (SECL)

	Project	Capacity	No. of D/L	CIL Plant No.	Date Of Commissioning	Cumulative worked Hrs. till 31.03.2013	Geo-mining conditions
1.	Bisrampur	30/88	2	EXC 052 EXC 083	03.11.1964 10.08.1967	226512 206459	Single thin seam at shallow depth
2.	Kurasia	10/70	1	EXC 516	14.09.1981	132814	Multi seam working with thin partings in between
3.	Dola+Rajnagar	10/70	1	EXC 643	03.10.1984	171032	Two thick seams with thin parting in between
4.	Jamuna	10/70 5/45	2	EXC 1049 EXC 188	24.12.1988 05.06.1977	133672 194590	Thin seam at shallow depth
5.	Dhanpuri	20/90 10/70	2	EXC 1310 EXC 927	10.04.1990 18.02.1987	140424 142227	Thin seam at shallow depth
Total for SECL			8				

Table 2.6- Northern Coalfields Ltd. (NCL)

	Project	Capacity	No. of D/L	CIL Plant No.	Date Of Commissioning	Cumulative worked Hrs. till 31.03.2013	Geo-mining conditions
1.	Amlohri	24/96	1	EXC	01.07.1993	120200	MOHER-SUB BASIN,

				1725			SIngrauli Coalfield. The NCL is presently working in Moher sub-basin of Singrauli coalfield. The basin has three seams in most of its area. The upper seams are 8-10 m thick with a parting of about 40 m in between. The lowermost seam is 16-22 m thick and has a parting of about 40 m between it and the second seam. The seams are flat (about 2 degree gradient). Upper seams are worked by shovel dumper combination and draglines are used only for removal of OB above the bottom most seam. When all the three seams are worked in any project of this sub-basin, the percentage of OB handled by dragline will only be 20-25 % of the total OB.
2.	Bina	24/96 24/96 10/70 10/70	4	EXC 1454 EXC 1455 EXC 1456 EXC 1457	10.03.1986 06.04.1987 15.05.1978 08.05.1979	165658 163712 203590 208512	
3.	Dudhichua	24/96 24/96 24/96 24/96	4	EXC 1393 EXC 1918 EXC 2156 EXC 2354	09.08.1991 16.08.1995 15.01.1999 10.10.2001	111623 102448 72804 70199	
4.	Jayant	24/96 24/96 24/96 15/90	4	EXC 554 EXC 655 EXC 1065 EXC 245	01.10.1983 03.03.1985 21.02.1989 01.04.1979	198825 186913 163165 195276	
5.	Khadia	20/90 20/90	2	EXC 1698 EXC 1849	02.10.1992 03.07.1995	125659 115165	
6.	Nigahi	24/96	4	EXC 2233	03.04.2002	71742	

		24/96		EXC 2270	15.04.2003	64741	
		20/90		EXC 1727	01.04.1992	126803	
		20/90		EXC 1761	25.03.1994	122229	
Total for NCL			19				

Total for Coal India Ltd. = 38

2 nos. of dragline are under commissioning in NCL & 4 more likely to be commissioned in near future in NCL.

2.9.2 Dragline in other mines

Table 2.7- Singareni collieries Co. Ltd. (SCCL)

	Project	Capacity of Dragline	No. of Draglines	Geo-mining conditions
1.	Ramagundam OC-I	24x96	1	Upper seams exposed by shovel-dumper. Lower seams exposed by dragline
2.	Ramagundam OC-III	30 cu.m	1	Parting between two seams taken by dragline
Total for SCCL			2	

Grand total for India = 40

Chapter 03

PARAMETER COLLECTION, RECORDING AND ACQUIRING

The parameters such as bucket capacity, boom length, reach, dumping height, cut width, angle of repose, highwall angle, bench height, digging depth, method of working of two draglines under study were collected and the parameters such as cycle time were recorded from the field. The parameters from Samaleswari (MCL) mine of scheduled shift hours, average working hours, average idle hours, average breakdown hours, and average maintenance hours were acquired from previous recorded data and those of Singareni OC-I mines was acquired from a previous field study by Rai., 2004. Details of these parameters can be found in chapter 04. Any relevant literature regarding these parameters can be found in chapter 02.

A case study of Rajnagar OC mines is performed and the parameters are mentioned in chapter 05.

Chapter 04

CALCULATION AND PROGRAMMING

4.1. Dragline balancing diagram

Balancing diagram can be defined as the graphical representation of the scheme to be adopted for determining the suitable seating position of the dragline in order to get maximum overburden accommodation in de-coaled area with least rehandling for achieving high rate of coal exposure and ensuring slope stability (Rai, 1997).

The balancing diagram assists in determining the coal exposed by a dragline, the percentage of overburden, rehandling and the volume of overburden to be accommodated in the de-coaled area (Singh and Rai, 1998).

Besides these, balancing diagram shows the dragline cuts and spoil geometry (in two dimensions) cross-section, height of dragline bench and cut width taken by the dragline. The cuts sequence by dragline, key cut (box cut), first cut (next to key cut), and first-dig can be estimated through the cross-sections drawn in diagram (Pundari, 1981).

4.2. Purpose of drawing balancing diagram

a) It shows the dragline cut sections i.e. key cut, first cut (next to key cut), first dig (next to first cut) and rehandled section (as per mode of operation).
b) It shows the dragline bench height, cut width taken by draglines, thickness of coal seam and gradient and various slope angles.
c) Determination of rate of coal exposure (daily/monthly or annually).
d) Calculation of workload distribution on each dragline in respect of their annual productivity (i.e. cross-section area taken by each dragline should be in the same ratio as their annual productivity).

e) Calculating the percentage of rehandling.
f) Calculating the overburden to be accommodated in the de-coaled area.

4.3. Preparation of Dragline balancing diagram

Let BCDE be the cross-sectional area to be removed to expose coal seam A B C O. for convenience, this area be called First-dig.

Let, A1= First-dig ……………… (1)

Now the dragline sitting on the highwall side removes the blasted overburden which lies in the cross-sectional area of first-dig. Maximum amount of overburden which can be accommodated in the dump FGKH is limited by the reach of the dragline and designed dump-slope.

Let A2= Dump Area ………….. (2)

Assuming S to be the swell factor of the overburden material, actual area of overburden required to be accommodated in dump would be A1S.

Let, A3= A1S-A2 …………….. (3)

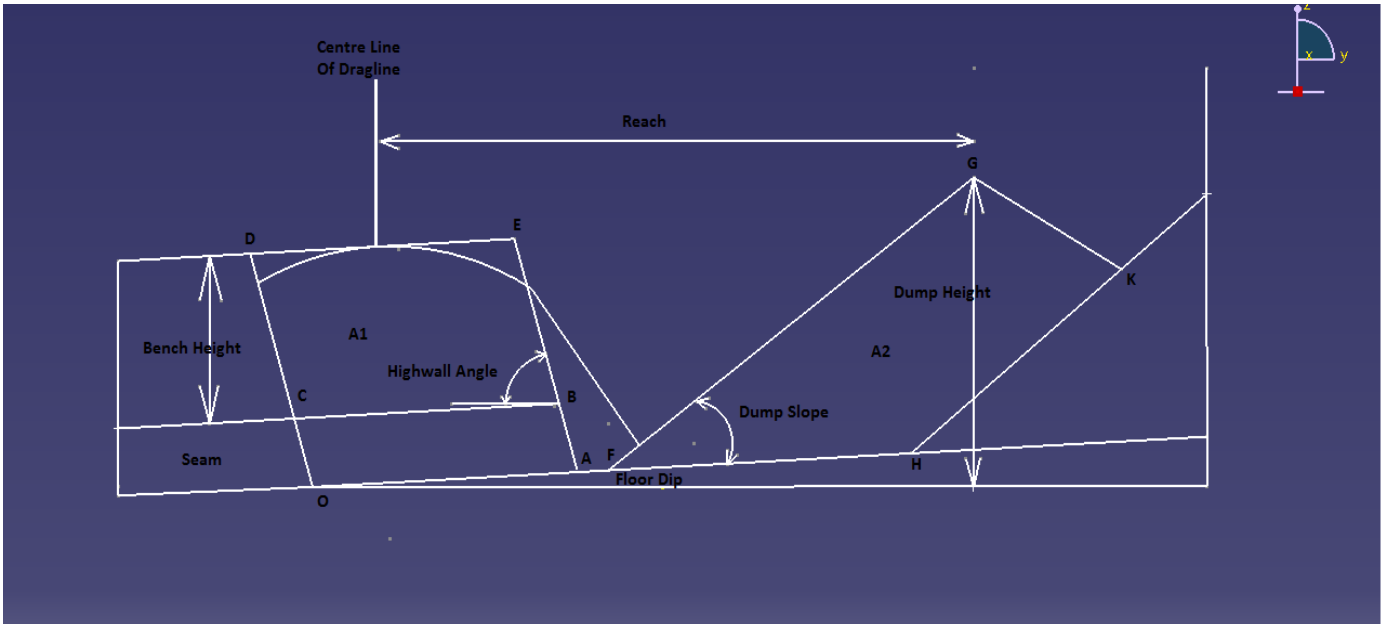


Fig 4.1: Dragline balancing diagram for total sidecasting

Case-1. When A3<0

In this case the dump area is incapable of accommodating overburden more than the available first-dig quantities. This implies that height of the dragline bench or cut width may be increased such that the first-dig quantity is increased. This process is repeated till the dump area is equal to the losses first dig quantity.

Case-2. When A3=0

This indicates an optimum solution for the simple side-casting method of dragline deployment. In simple sidecasting operation there is no rehandling of material and thereby it is the most economical operation. Any increase in the height of dragline bench or cut width would give rise to an increase in first-dig and this increase is not possible to be accommodated in the dump.

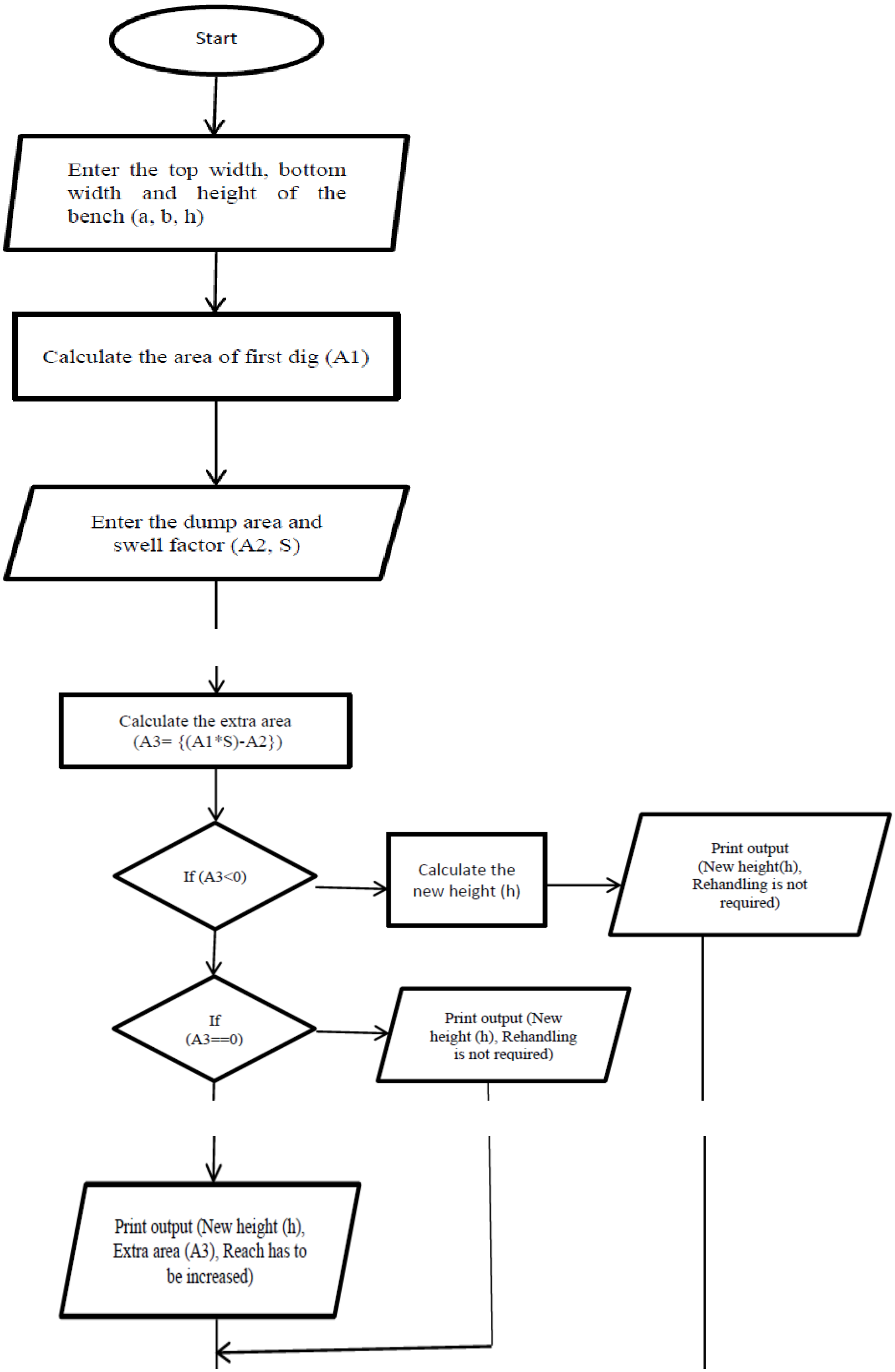
Case-3. When A3>0

This implies that the dump is incapable of taking the loose first-dig completely and A3 amount of overburden would be left as residual. This residual can be handled in two ways, either by transporting and dumping elsewhere or by generating extra dump capacity can be increased by increasing reach. Reach can be increased by selecting different equipment with higher reach. But the choice of availability is limited. Alternatively the reach can be increased by shifting the dragline towards the dump side. Extended bench method of dragline deployment is employed for this purpose.

4.4. Developing a computer based program (in MATLAB) on dragline bench designing (keeping the cut-width constant) and determine whether or not rehandling is required.

```
function area1()
        global A1 a b h;
        A1=(((a+b)/2)*h);
    display('the first dig area is now ');
   display(A1);
function area3()
        global A3 A1 s A2;
        A3=(A1*s-A2);
 global a b h A2 s A3 A1 h1;
     a=input('Enter length');
     b=input('Enter breadth');
     h=input('Enter height');
area1();
    A2=input('Enter dump area');
    s=input('Enter swell factor');
area3();
     if(A3<0)
display('the height earlier was');
```

```
display(h);
                    h=((2*A2)/(s*(a+b)));
            display('the height is now');
         display(h);
             display('the extra area is zero');
           display('no rehandling is required');
  end
           if(A3==0)
display('the height is now');
           display(h);
         display('no rehandling required');
end
        if(A3>0)
display('the height is now' );
display(h);
             display('and the extra area is');
           display(A3);
        display('the reach has to be increased');
end
```


Start
Enter the top width, bottom width and height of the bench (a, b, h)
Calculate the area of first dig (A1)
Enter the dump area and swell factor (A2, S)
Calculate the extra area (A3= {(A1*S)-A2})
If (A3<0)
Calculate the new height (h)
Print output (New height(h), Rehandling is not required)
If (A3==0)
Print output (New height (h), Rehandling is not required)
Print output (New height (h), Extra area (A3), Reach has to be increased)

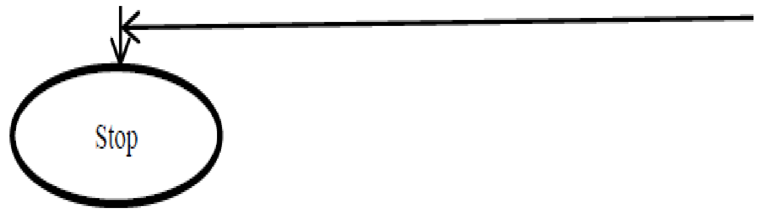


Fig 4.2: Flowchart for the dragline bench designing (keeping the cut-width constant) and determine whether or not rehandling is required

4.5. Output (using user-defined data)

4.5.1 Case 1 (A3<0)

Input entered:

Top width of cut (length):	80 m
Bottom width of cut (breadth):	50 m
Height of bench:	28m
Dump area:	2400 m2
Swell factor:	1.25

Output:

The height earlier was 28.00 m

The new height is 29.538 m

The extra area is 0

No rehandling is required

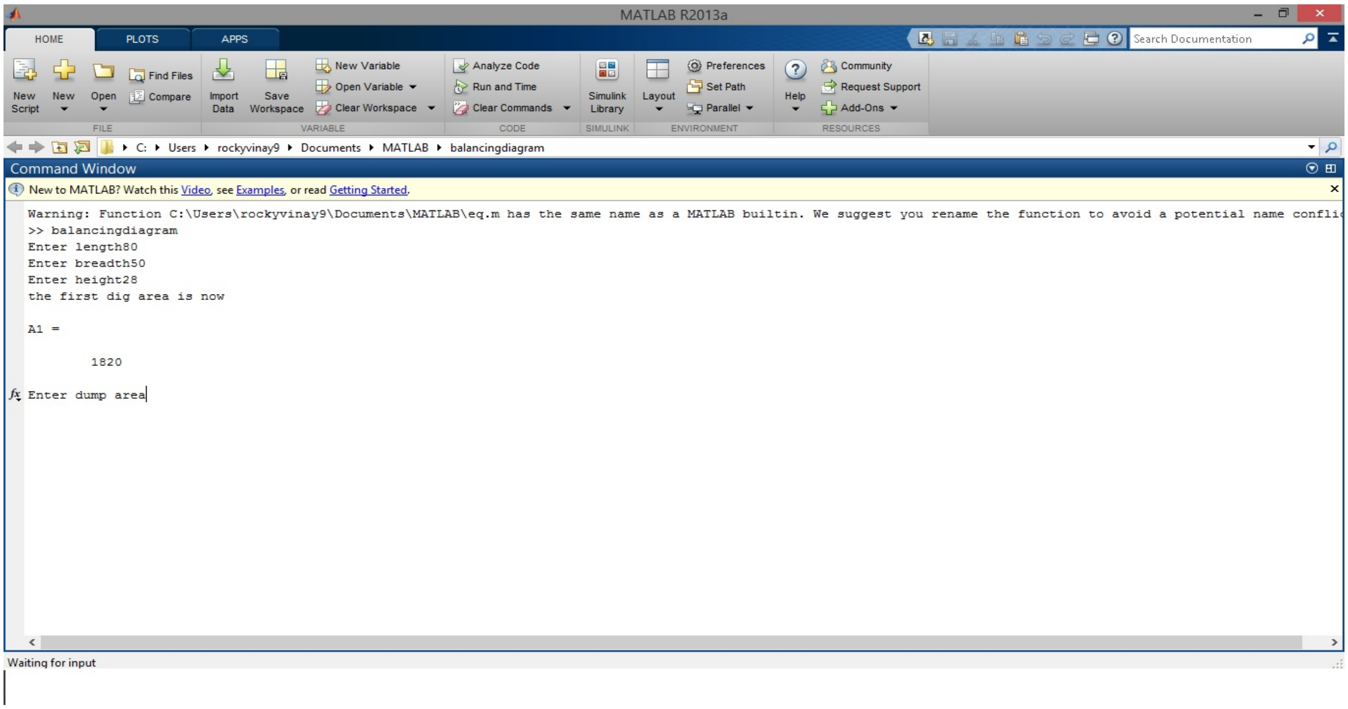


Fig 4.3: Output screenshot 1

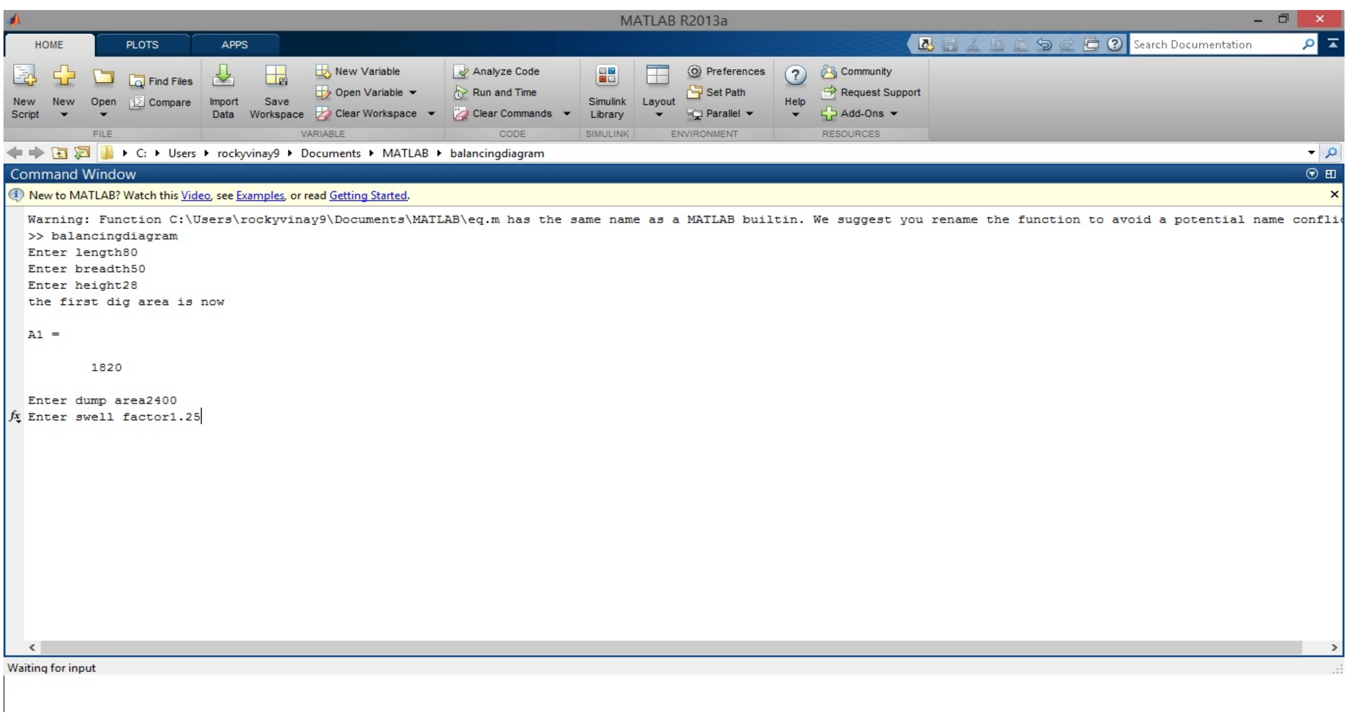


Fig 4.4: Output screenshot 2

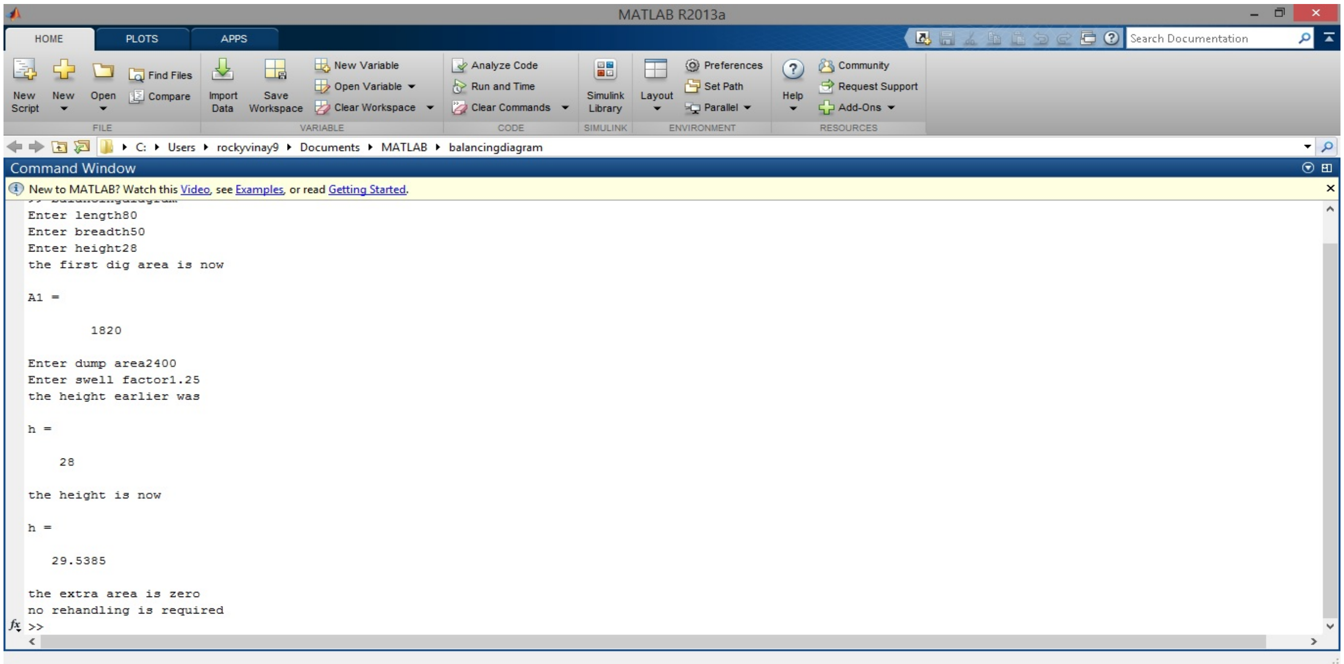


Fig 4.5: Output screenshot 3

4.5.2. Case 2 (A3=0)

Input entered:

Top width of cut (length):	80 m
Bottom width of cut (breadth):	50 m
Height of bench:	28m
Dump area:	2275 m2
Swell factor:	1.25

Output:

The new height is 28.00 m

No rehandling is required

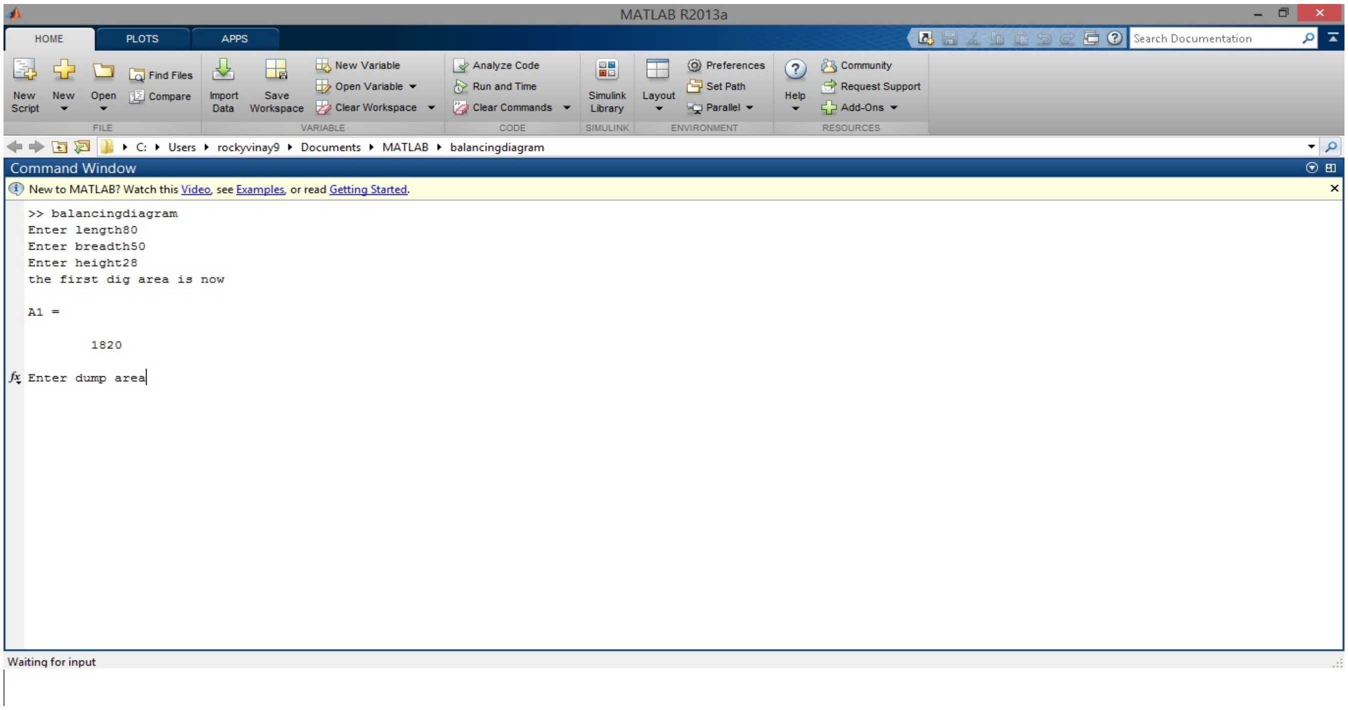


Fig 4.6: Output screenshot 4

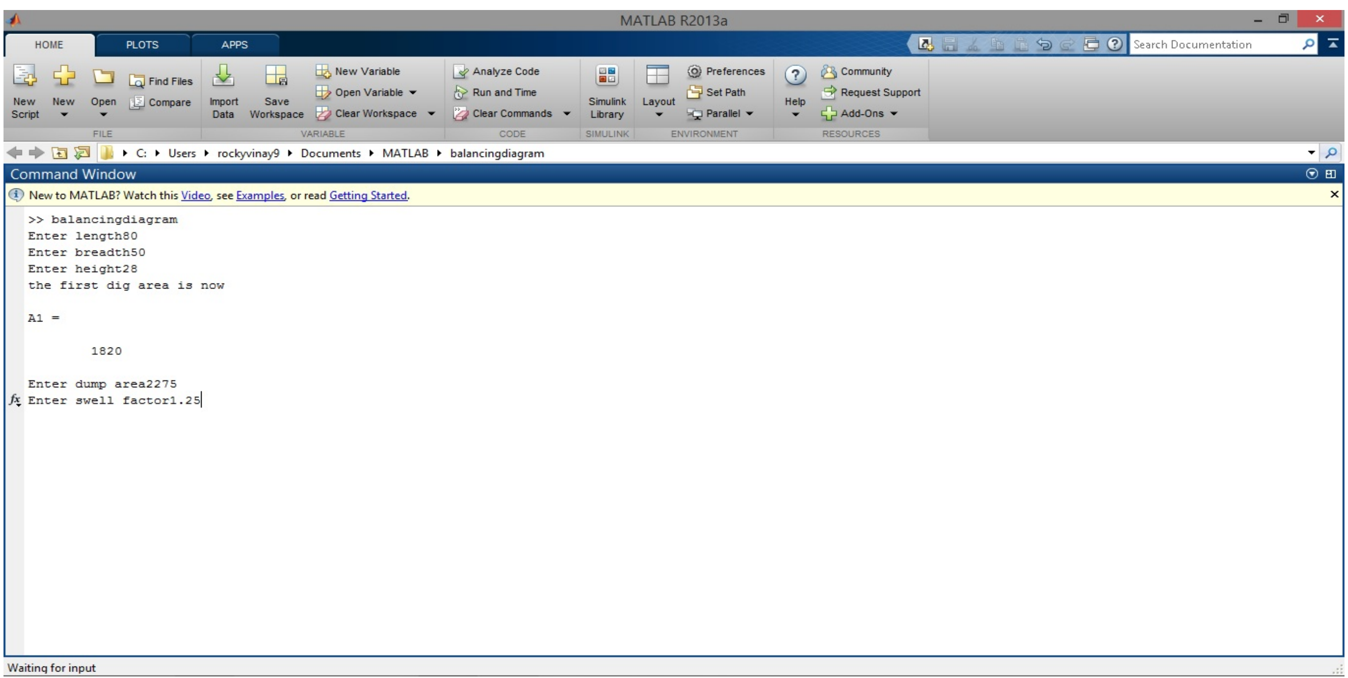


Fig 4.7: Output screenshot 5

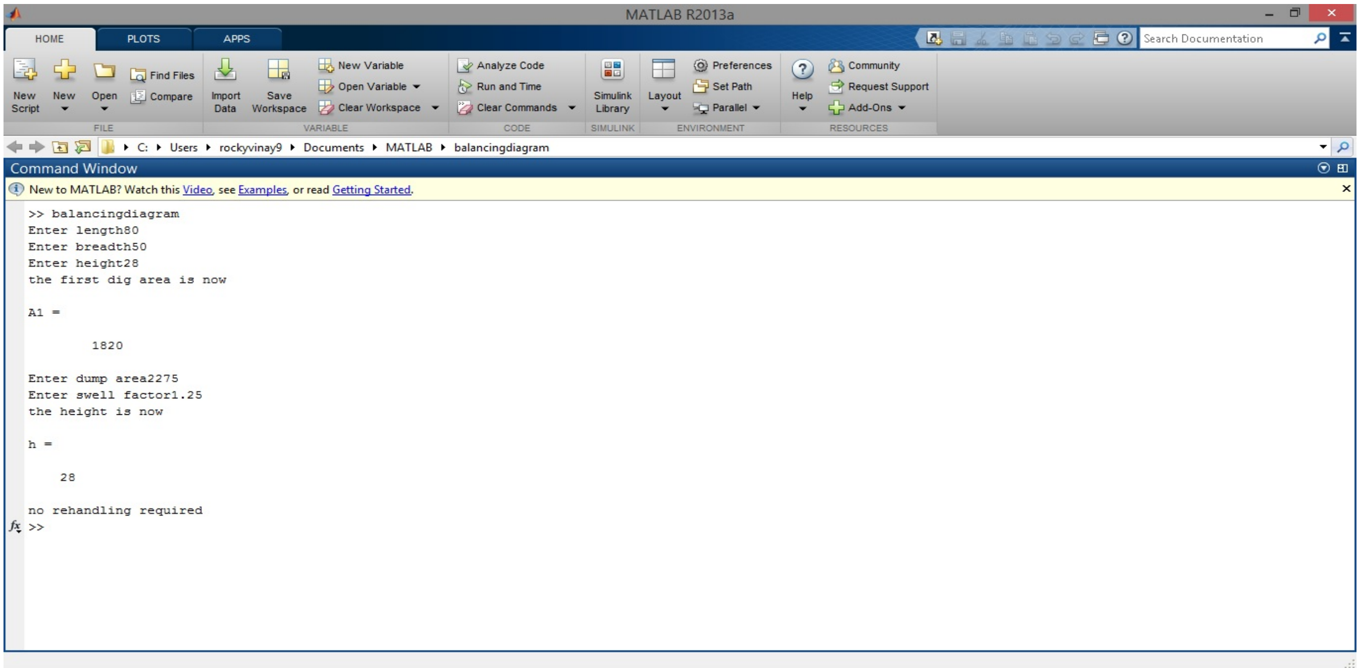


Fig 4.8: Output screenshot 6

4.5.3. Case3 (A3>0)

Input entered:

Top width of cut (length):	80 m
Bottom width of cut (breadth):	50 m
Height of bench:	28 m
Dump area:	2000 m2
Swell factor:	1.25

Output:

The height now is 28.00 m

The extra area is 275 m2

Reach has to be increased

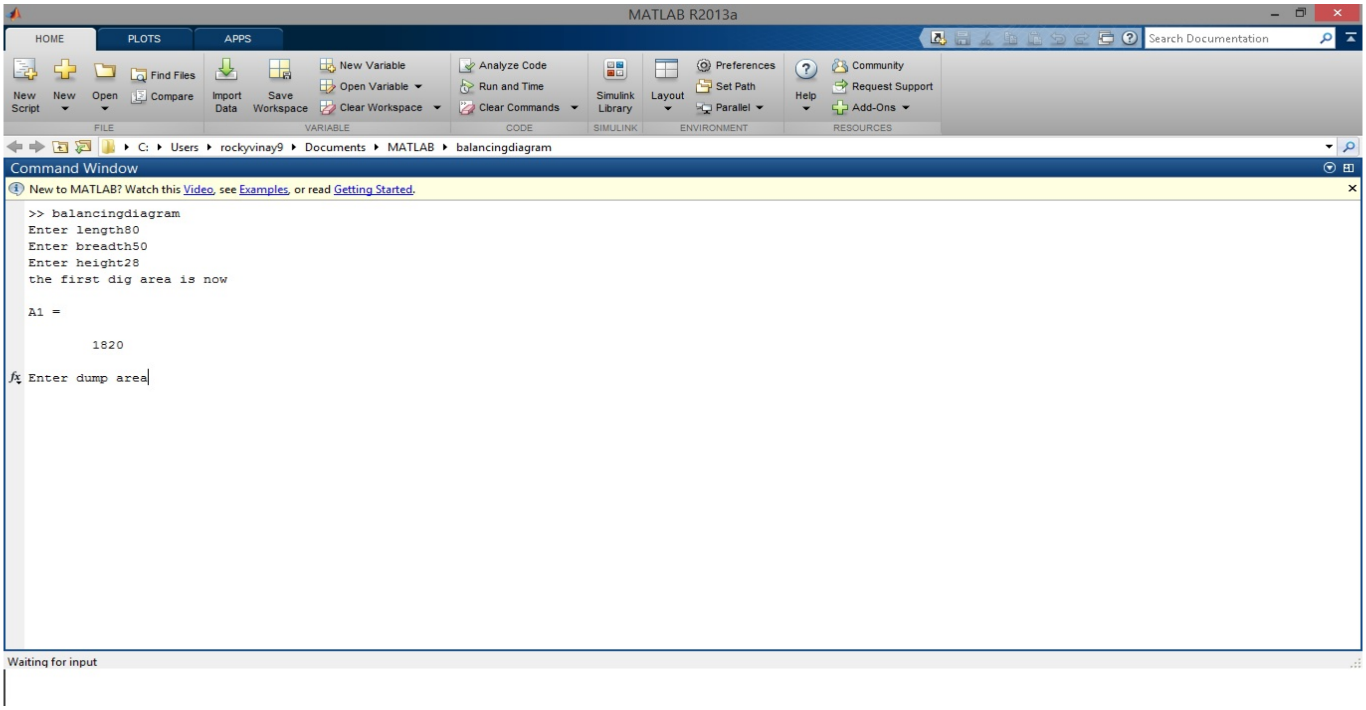


Fig 4.9: Output screenshot 7

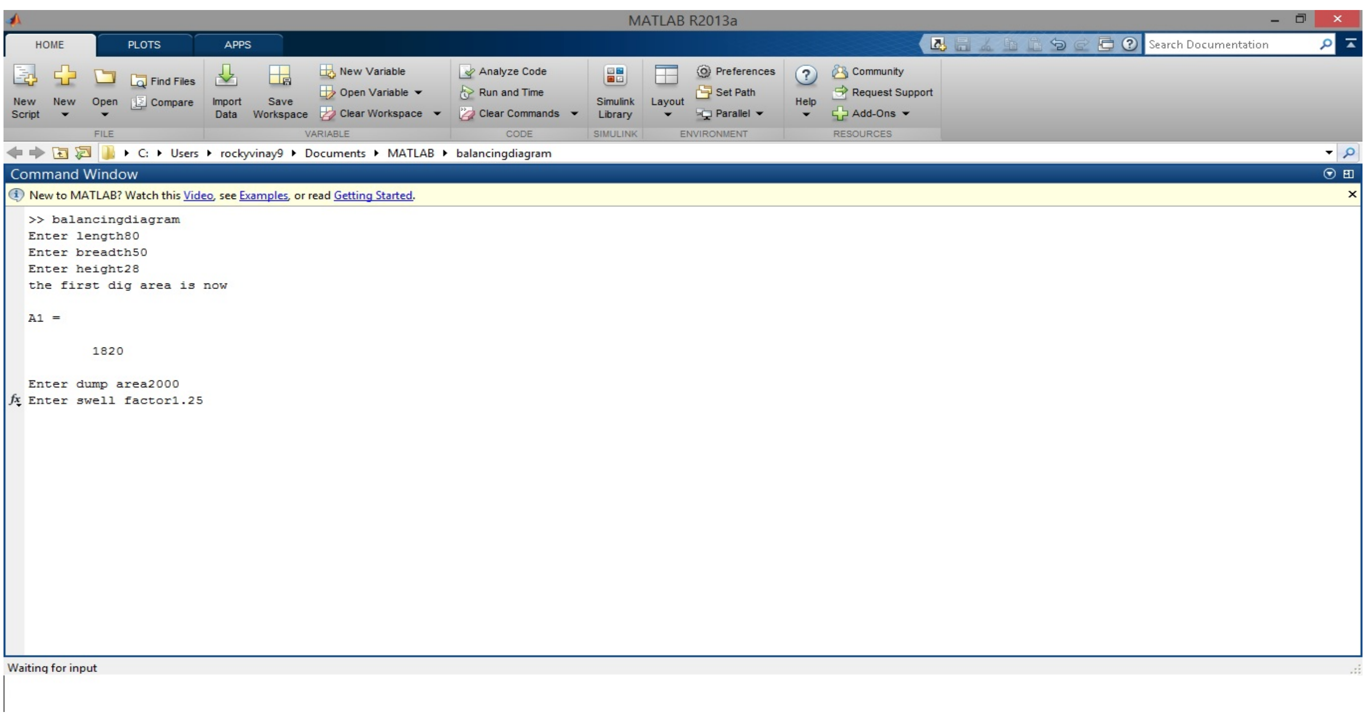


Fig 4.10: Output screenshot 8

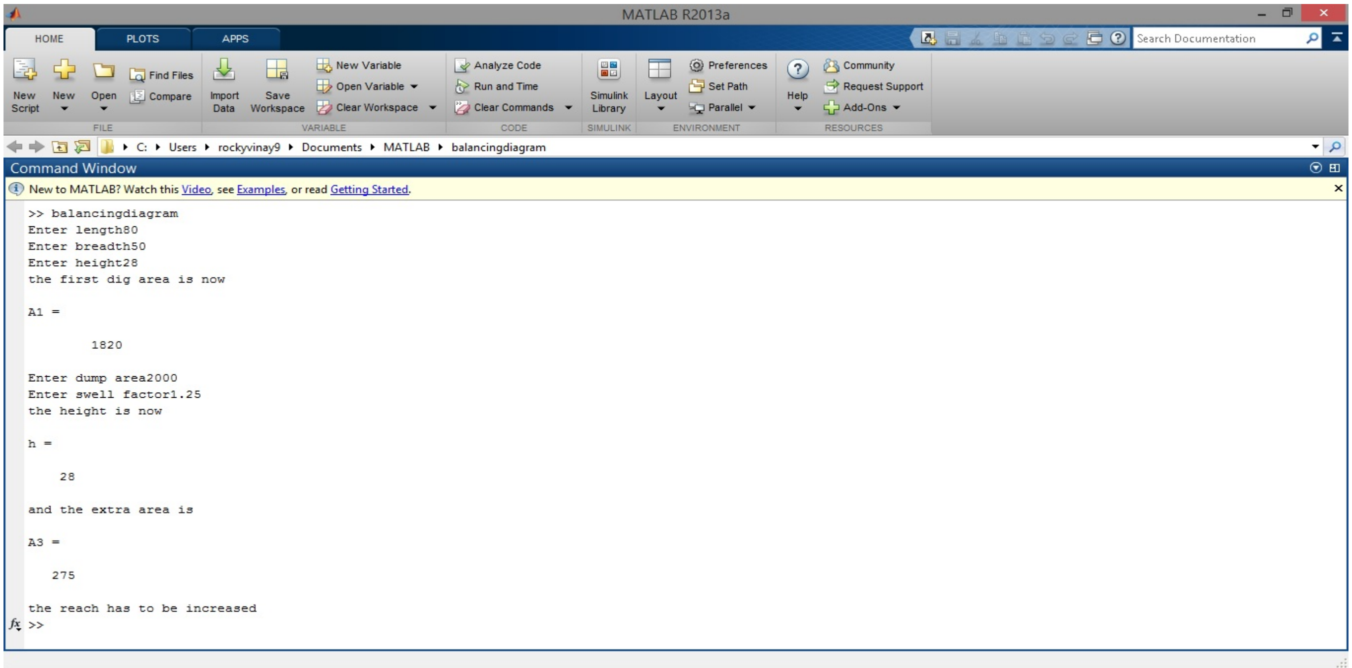


Fig 4.11: Output screenshot 9

4.6. Projection of annual output, calculation of ownership, operating cost and cost per tonne of coal exposed by dragline

4.6.1. Evaluation of Availability and Utilization

To evaluate the Availability (A) and Utilization (U) the field data acquired was substituted in the Eqs (i) and (ii).

A= [SSH – (MH+BH)]/ SSH. . . . .(i)

U = [SSH – (MH+BH+IH)]/SSH . . . . (ii)

Where,

SSH is a scheduled shift hour,

MH is maintenance hours,

BH is breakdown hours,

IH is idle hours

Based on the observed and recorded data in terms of average cycle time, A and U values the annual output (P1) of the dragline has been projected using Eq (iii):

$$P1 = (B/C)*A*U*S*F*M*Ns*Nh*Nd*3600 \quad \text{(iii)}$$

Where,

B is bucket capacity of the dragline in cubic meter,

C is the average total cycle time of dragline in second,

S is the swell factor,

F is the fill factor,

M is the machine travelling and positioning factor,

Ns is the number of operating shifts in a day,

Nh is the number of operating hours in a shift,

Nd is the number of operating days in a year,

In the above equation the values of average cycle time (C), A and U were substituted as per the recorded and acquired field observations. Remaining factors in the Eq (iii) (S, F, M, Ns, Nh, Nd) were substituted as per the recommendations made by CMPDI in regard to the values of these factors in Indian coal mines. The suggested values for these factors are given in Table 1.

Table 4.1 – Productivity factors for dragline as per CMPDI recommendations

Particulars	Recommended values
Swell factor (S)	0.719
Fill factor (F)	0.733
Machine travel and positioning factor (M)	0.8
No. of shifts in a day (Ns)	3
No. of hours in a day (Nh)	8
No. of days in a year (Nd)	365

Table 4.2 – Parameters of Singareni OCP – I Dragline

SL NO.	Parameters	Details
1.	Dragline(bucket(m3)/boom(m))	24/96
2.	Make	Rapier & Ransom (England)
3.	Max operating radius (m)	88
4.	Bench height (m)	30-35
5.	Cutting width (m)	60
6.	Highwall slope (degrees)	70
7.	Bench slope (degrees)	60
8.	Angle of repose (degrees)	38
9.	Digging depth (m)	25
10	Reach of dragline (m)	73
11.	Method of working	Extended Bench method
12.	Thickness of coal seam (m)	4.5
13.	Scheduled shift hours (SSH)	720
14.	Working hours (WH)	507
15.	Maintenance hours (MH)	119
16.	Breakdown hours (BH)	33
17.	Idle hours (IH)	61
18.	Standard cycle time (s)	60
19.	Observed cycle time (s)	61.7

Table 4.3 – Parameters of Samaleswari Dragline

Sl. NO.	Parameters	Details
1.	Dragline(bucket(m3)/boom(m))	10/70
2.	Make	Russian
3.	Max operating radius (m)	58
4.	Bench height (m)	35-40
5.	Cutting width (m)	45
6.	Highwall slope (degrees)	70
7.	Bench slope (degrees)	60
8.	Angle of repose (degrees)	38
9.	Digging depth (m)	36
10	Reach of dragline (m)	58
11.	Method of working	Simple side casting
12.	Thickness of coal seam (m)	25
13.	Scheduled shift hours (SSH)	720
14.	Working hours (WH)	540
15.	Maintenance hours (MH)	90
16.	Breakdown hours (BH)	30
17.	Idle hours (IH)	60
18.	Standard cycle time (s)	60
19.	Observed cycle time (s)	66.3

4.6.2. THE MAXIMUM DEPTH THAT CAN BE WORKED BY A DRAGLINE IS GIVEN BY THE FORMULA:

$$H = \{t + \tan x\,(R - W/4)\} / \{S + (\tan x / \tan y)\} \quad \text{(iv)}$$

Where,

H is the maximum depth that can be worked by the dragline.

T is the thickness of coal seam.

X is the angle of repose of overburden.

R is the reach of dragline.

S is the swell factor.

W is the width of cut.

Y is the slope angle of highwall to horizontal.

4.6.3. AMOUNT OF REHANDLE (PRM):

$$Prm = [(1.125\,t + 0.684\,H + 0.1\,R)/W] + [(0.25\,t2 - 0.4\,Rt - 0.16\,R2)/HW] + [(0.1\,t + 0.08\,R - 0.01W)/H] \quad \text{(v)}$$

Where,

Prm is the % amount of rehandle material.

H is the overburden dump height.

T is the thickness of coal seam.

R is the reach of dragline.

W is the width of cut.

By using the recorded, acquired data and recommended values in the Eqs (iii) (iv) & (v), we get:

For OCP – I Dragline (24/96):

1. The projected annual output of the Singareni OCP-I dragline is 2.807 M cu.m.

2. The maximum depth that can be worked = 29.74 m.

3. The amount of rehandle percentage = 40.2.

For Samaleswari Dragline (10/70):

1. The projected annual output of the Samaleswari dragline is 1.253 M cu.m.

2. The maximum depth that can be worked = 36.74 m.

3. The amount of rehandle percentage = 0.

4.6.4. Calculation of ownership and operating cost of dragline (Singareni OC-I dragline)

A. Cost of ownership per year of the 24/96 dragline

(i) Cost of equipment

Cost of the 24/96 dragline = Rs. 1000 million

(ii) Depreciation cost for 25 year i.e. annual flat rate of 4%

Annual depreciation cost of 24/96 dragline = Rs. 40 million

(iii) Annual cost of ownership (24/96)

Average annual investment = [(N+1)/2N] * cost of dragline

Where N = Life of dragline; = Rs. [(1000*26)/2*25] million = Rs. 520 million

(iv) Annual interest, insurance rates and taxes i.e. annual flat rate of 12.5%

= 15 % of Rs. 520 million

= Rs. 78 million

Hence the total ownership cost per year = (ii) + (iv)

= Rs. (40+78) million

= Rs. 118 million

B. Operating cost per year of the 24/96 dragline

(i) Annual manpower cost (salary and wages)

Operator cost @ Rs. 0.20 millions/operator for 2 operators in 3 shifts = Rs. 1.20 million

Helper cost @ Rs. 0.14 million for 1 operator in 3 shifts = Rs. 0.42 million

Total manpower cost = Rs. 1.62 million

(ii) Annual power and energy consumption on the basis of 13.65 MKWH for 24/96

Annual power consumption cost @ Rs. 4.89/KWH = Rs. 4.89*13.65*106

= Rs. 66.75 million

(iii) Annual lubrication cost @ 30% of power consumption = Rs. 20.025 million

(iv) Annual maintenance cost @ 20% of depreciation cost = Rs. 8 million

Major breakdown cost @ 2% of cost of equipment = Rs. 20 million

Total maintenance cost = Rs. 28 million

Hence, Total Annual operating cost = Manpower cost/year + Electrical cost/year + Maintenance cost/year + Lubrication cost/year

= Rs. (1.62 + 66.75 + 20.025 +28) million

= Rs. 116.4 million

Total ownership and operating cost

= ownership cost/year + operating cost/year

= Rs. (118+116.4) million

= Rs. 234.4 million

Dragline operating cost per m3 overburden handle considering annual output of 24/96 as 2.807M cu.m = Rs. [(234.4*106)/(2.87*106)]= Rs. 81.67

4.6.5. Calculation of cost per ton of coal exposed by Singareni OCP – I Dragline by extended bench method

Dragline deployed is 24/96 having a production capacity of 2.87 M cu.m/year

Percentage rehandling is 40.2%

Total overburden handled = overburden directly over the exposed coal + overburden rehandled

= overburden directly over the exposed coal (1+coefficient of rehandling)

Here, coefficient of rehandling = O.B rehandle/O.B removal to expose coal

Therefore, 2.87 M cu.m = overburden directly over the exposed coal *1.40

Hence overburden directly over the exposed coal removed by the dragline = (2.87/1.40)M cu.m

= 2.05 M cu.m

Amount of coal exposure = (2.05 M cu.m)/ (4.2 m3/te)

= 0.82 Mte

Estimated cost/tonne of coal exposed = Rs. (234.34*106)/ (0.82*106)

= Rs. 285.78

= Rs. 285.78 per te of coal exposed.

4.6.6. Calculation of ownership and operating cost of dragline (Samaleswari dragline)

A. Cost of ownership per year of the 10/70 dragline

(i) Cost of equipment

Cost of the 24/96 dragline = Rs. 300 million

(ii) Depreciation cost for 25 year i.e. annual flat rate of 4%

Annual depreciation cost of 10/70 dragline = Rs. 12 million

(iii)Annual cost of ownership (10/70)

Average annual investment = [(N+1)/2N] * cost of dragline

Where N = Life of dragline; = Rs. [(300*26 million)/ (2*25)] = Rs. 156 million

(iv) Annual interest, insurance rates and taxes i.e. annual flat rate of 12.5%

= 15 % of Rs. 156 million
= Rs. 23.4 million

Hence the total ownership cost per year = (ii) +(iv)
= Rs. (12+23.4) million
= Rs. 35.4 million

B. Operating cost per year of the 10/70 dragline

(i) Annual manpower cost (salary and wages)

Operator cost @ Rs. 0.20 millions/operator for 2 operators in 3 shifts = Rs. 1.20 million

Helper cost @ Rs. 0.14 million for 1 operator in 3 shifts = Rs. 0.42 million

Total manpower cost = Rs. 1.62 million

(ii) Annual power and energy consumption on the basis of 9.07 MKWH for 24/96

Annual power consumption cost @ Rs. 4.89/KWH = Rs. $4.89*9.07*10^6$

= Rs. 44.35 million

(iii) Annual lubrication cost @ 30% of power consumption = Rs. 13.3 million

(iv) Annual maintenance cost @ 20% of depreciation cost = Rs. 2.4 million

Major breakdown cost @ 2% of cost of equipment = Rs. 6 million

Total maintenance cost = Rs. 8.4 million

Hence, Total Annual operating cost = Manpower cost/year + Electrical cost/year + Maintenance cost/year + Lubrication cost/year

= Rs. (1.62 + 44.35 + 13.3 +8.4) million

= Rs. 67.67 million

Total ownership and operating cost = ownership cost/year + operating cost/year

= Rs. (35.4+67.67) million

= Rs. 103.07 million

Dragline operating cost per m3 overburden handle considering annual output of 10/70 as 1.253 M cu.m = (Rs. 103.07*106)/ (1.253*106**)**

= Rs. 82.2

4.6.7. Calculation of cost per ton of coal exposed by Samaleswari Dragline by simple side casting method

Dragline deployed is 10/70 having a production capacity of 1.253 M cu.m/year

Amount of overburden handled = 1.253 M cu.m

Amount of coal exposure = (Annual production of dragline **/** Average stripping ratio)

= (1.253 M cu.m) **/ (**3 cu.m/te)

= 0.417 M te

Estimated cost/tonne of coal exposed = (Rs. 103.07 /te) / 0.417

= Rs. 247.17 /te of coal exposed

4.6.8. Developing a computer based program (in MATLAB) for projection of annual production of overburden, calculation of ownership, operating cost and cost per cu. m overburden handle of the dragline.

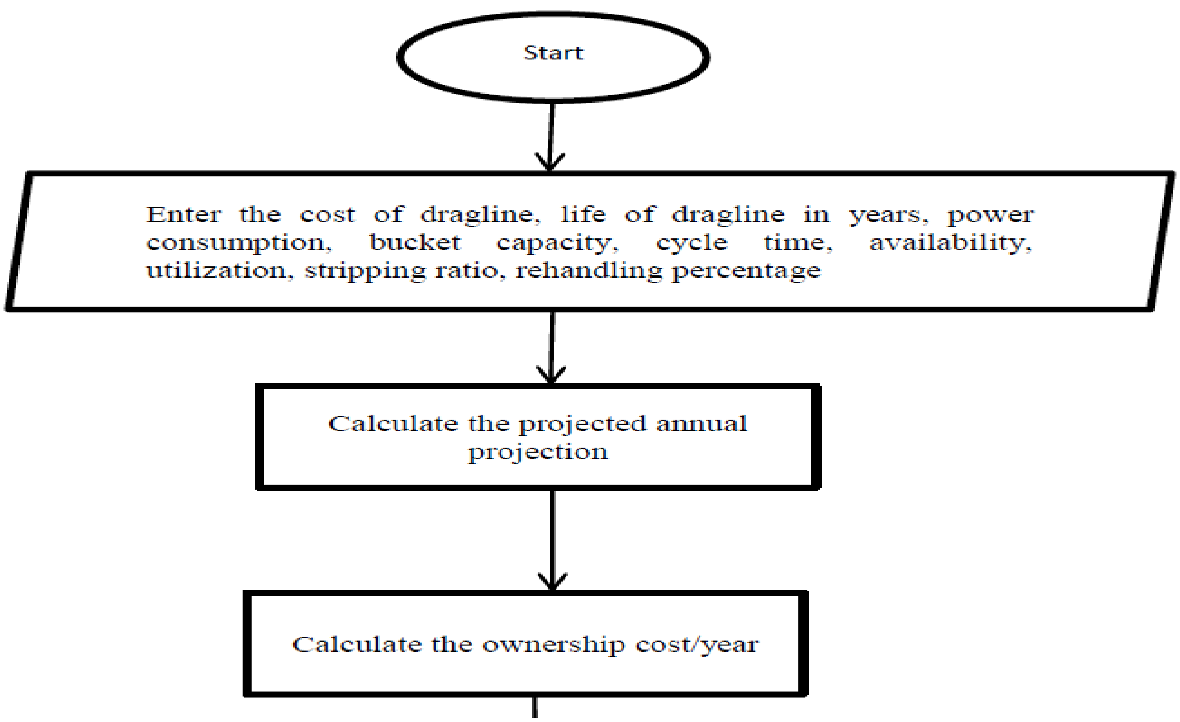
```
function annualproduction()
        format long g
        global b ct a ut;
    global pa;
            pa=(b/ct)*a*ut*0.719*0.733*0.8*8*3*365*60*60;
      display('the annual production of the dragline is ');
    display(pa);
function ownershipcost()
       format long g
```

```
global c n ;
global oc;
d=0.04*c;
aai=((n+1)/(2*n))*c;
ai=(15.00/100)*aai;
oc=d+ai;
display('the ownership cost of the ownership is');
display(oc);
function operatingcost()
format long g
global p d c;
global op;
d=0.04*c;
global op;
mc=(0.20*6.00)+(0.14*3.00);
ap=4.89*p*1000000;
al=0.30*ap;
am=(0.20*d)+(0.02*c);
op=mc+ap+al+am;
display('the operating cost of the dragline is');
display(op);
```

```
function totalcost()
            format long g
           global oc op;
           global ta;
                ta=oc+op;
       display('the total cost of the dragline is ');
      display(ta);
function overburdencost()
              format long g
              global ta pa;
             global ob;
                     ob=(ta/pa);
        display('the operating cost per cu.m overburden handle of the dragline is ');
      display(ob);
function coalexposure1()
               format long g
              global pa s ta;
                     ce=pa/s;
                     ec=ta/ce;
         display('the cost per tonne of coal exposed by the dragline is ');
       display(ec);
```

```
function coalexposure2()
        format long g
        global pa r s ta;
             to=(pa)/(1.00+r);
             tce=(to/s);
             tec=(ta/tce);
      display('the cost per tonne of coal exposed by the dragline is ');
    display(tec);
format long g
   global c n p b ct a ut s r t d;
   c=input('enter cost of dragline\n');
   n=input('no. of years\n');
   p=input('power consumption\n');
   b=input('bucket capacity\n');
   ct=input('cycle time\n');
   a=input('availability\n');
   ut=input('utilization\n');
   s=input('stripping ratio\n');
   r=input('rehandling in decimals\n');
   t=input('Enter type of method(simple sidecasting(s) or extended benchmethod(e))::','s');
 annualproduction();
```

```
ownershipcost();

operatingcost();

totalcost();

overburdencost();

            if(t=='s')

coalexposure1();

   end

            if(t=='e')

coalexposure2();

   end
```



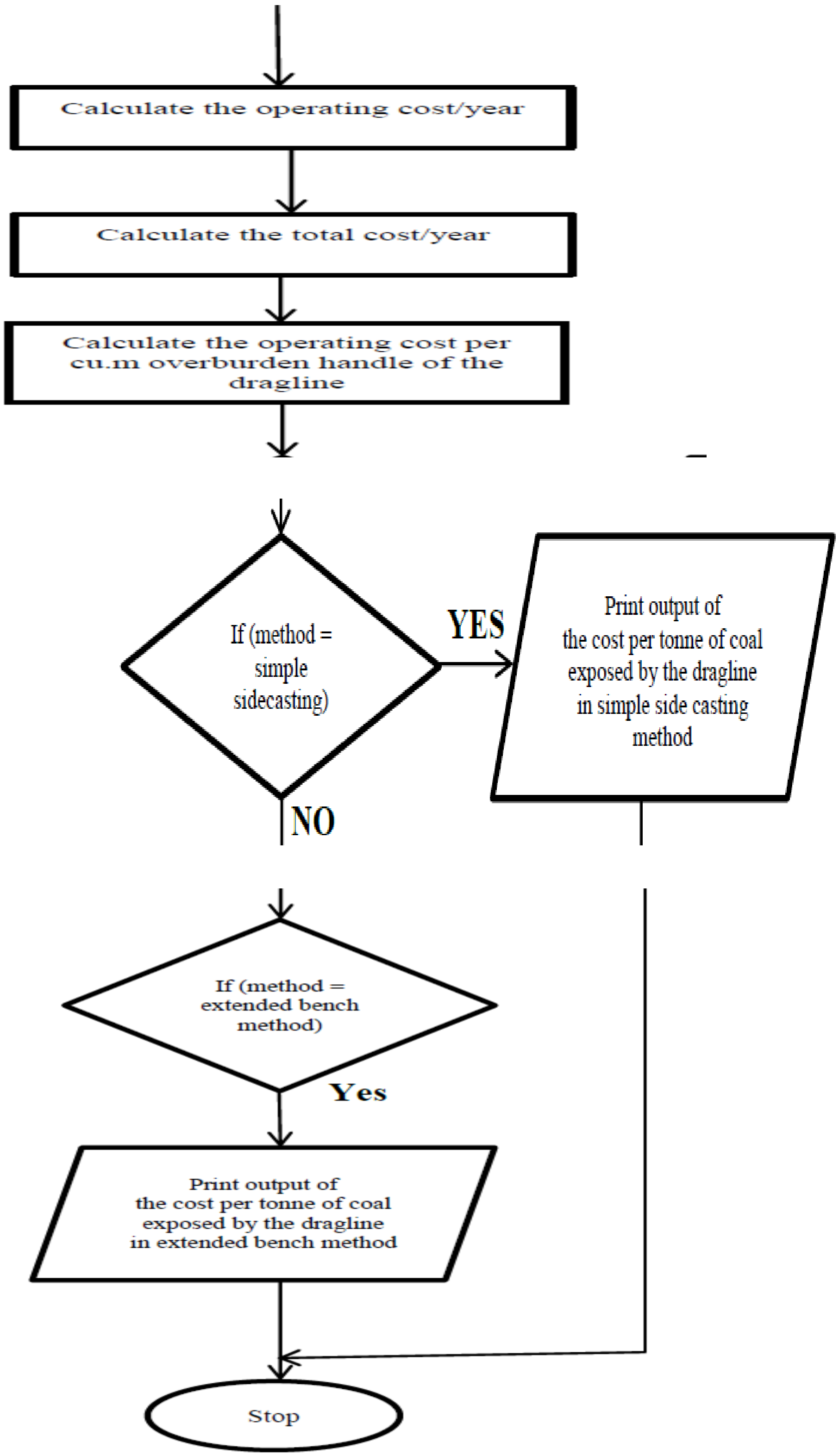


Fig 4.12: Flowchart for the dragline bench designing (keeping the cut-width constant) and determine whether or not rehandling is required

Input entered (For extended bench method):

Cost of dragline:	Rs. 1000 million
No. of years:	25
Power consumption:	13.65 KWh
Bucket capacity:	24 cu.m
Cycle time:	61.7 s
Availability:	0.7888
Utilization:	0.7041
Stripping ratio:	2.5
Percentage rehandling:	0.40

To enter the type of method (simple sidecasting (s) or extended bench method (e)): e

Output:

The annual production of dragline = 2.87 M cu.m

The ownership cost of dragline/year = Rs. 118 million

The operating cost of dragline/year = Rs. 114.7 million

The total costs of dragline/year = Rs. 232.7 million

The operating cost per cu.m overburden handle = Rs. 81.03

The cost per tonne of coal exposed by the dragline = Rs. 284.76

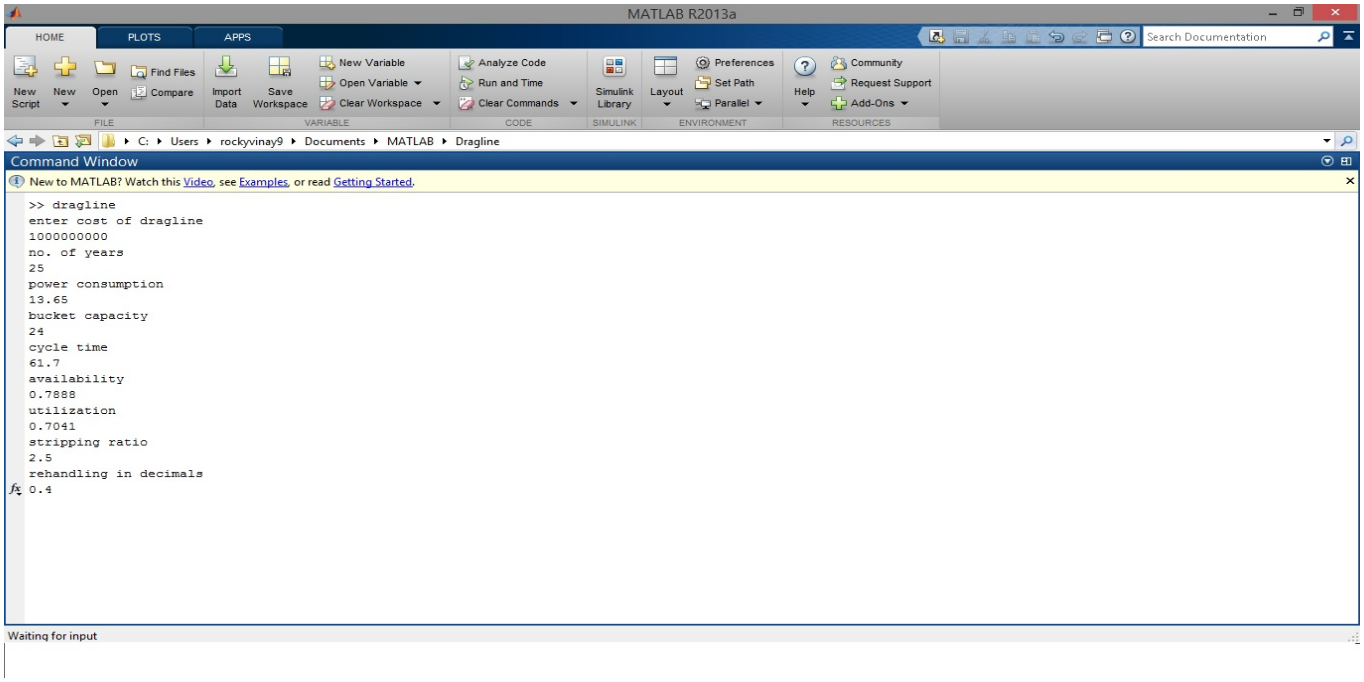


Fig 4.13: Output screenshot 10

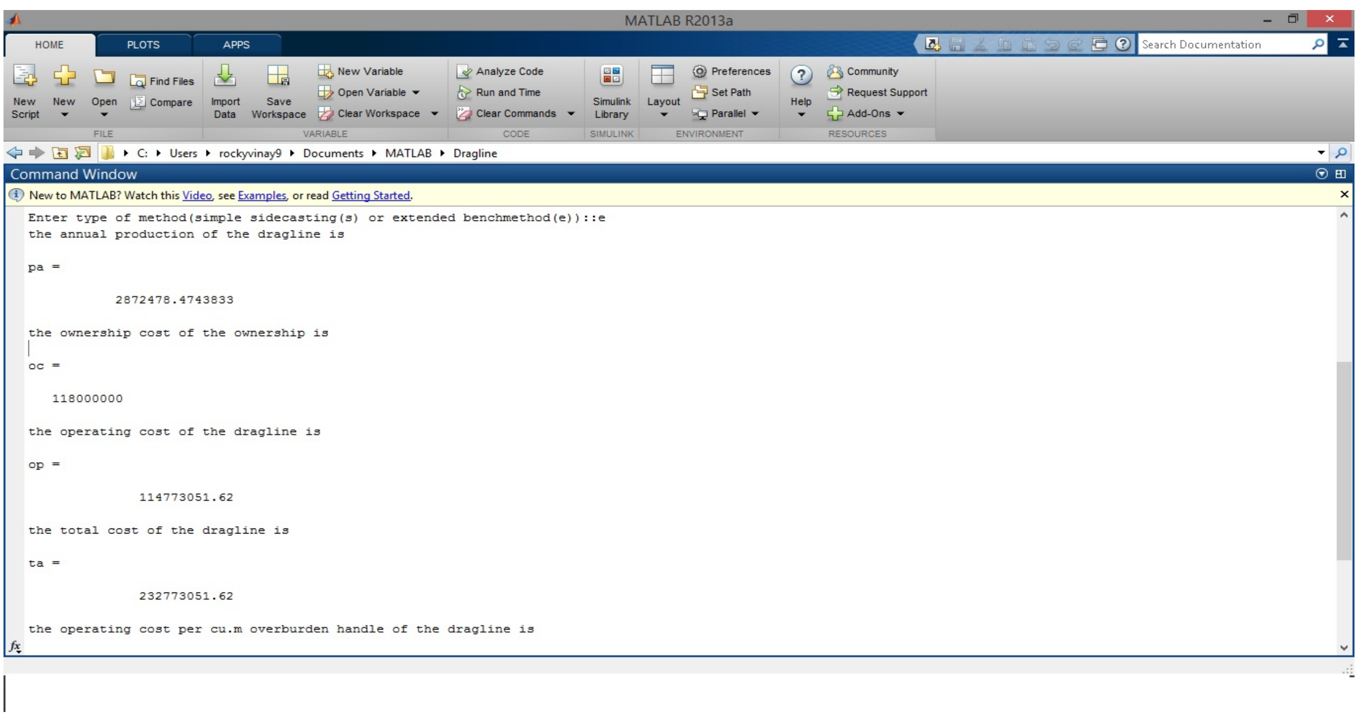


Fig 4.14: Output screenshot 11

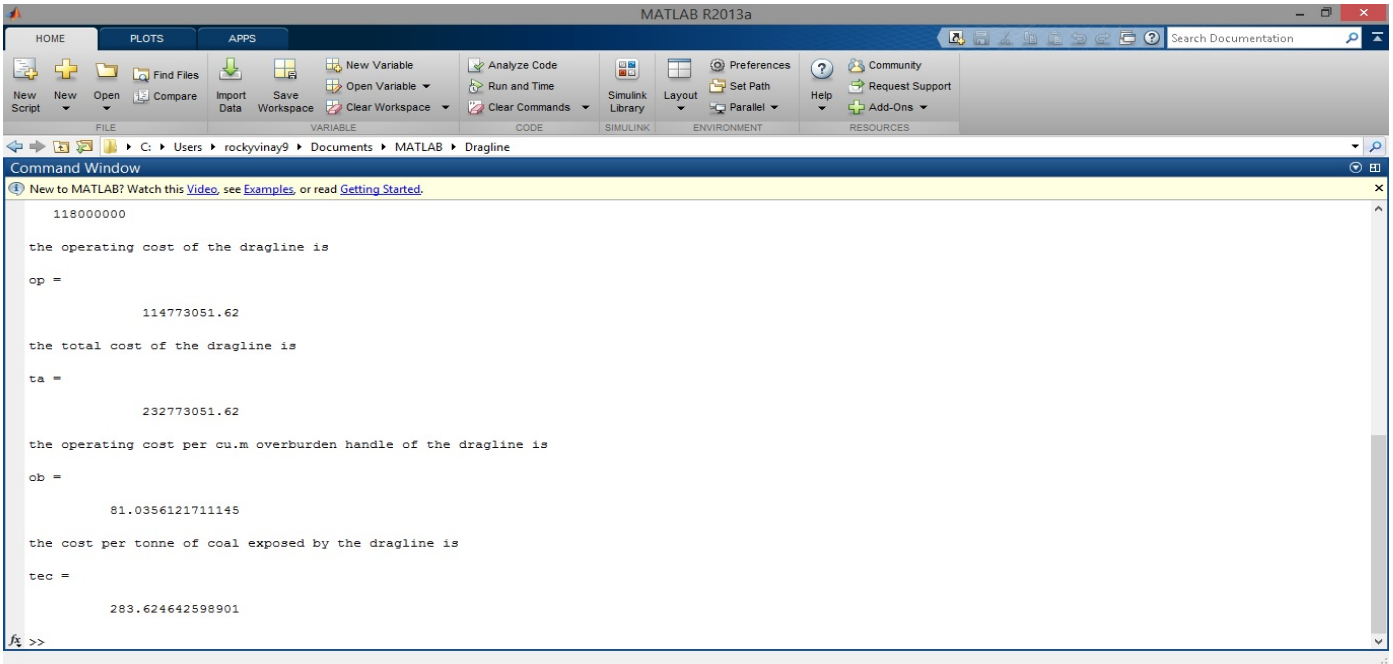


Fig 4.15: Output screenshot 12

Input entered (For simple sidecasting method):

Cost of dragline:	Rs. 300 million
No. of years:	25
Power consumption:	9.07 KWh
Bucket capacity:	10 cu.m
Cycle time:	66.3 s
Availability:	0.8333
Utilization:	0.75
Stripping ratio:	3

Percentage rehandling: 0

To enter the type of method (simple sidecasting (s) or extended bench method (e)): s

Output:

The annual production of dragline = 1.24 M cu.m

The ownership cost of dragline/year = Rs. 35.4 million

The operating cost of dragline/year = Rs. 66.05 million

The total costs of dragline/year = Rs. 101.45 million

The operating cost per cu.m overburden handle = Rs. 81.46

The cost per tonne of coal exposed by the dragline = Rs. 244.4

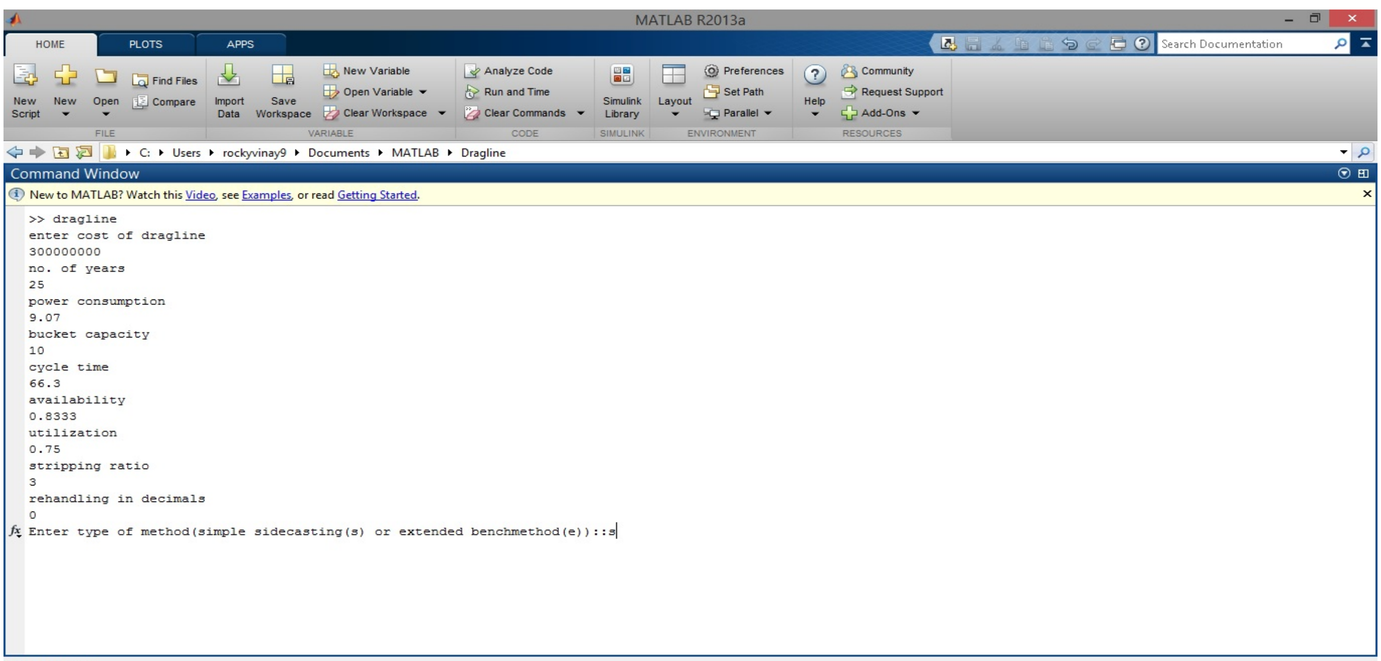


Fig 4.16: Output screenshot 13

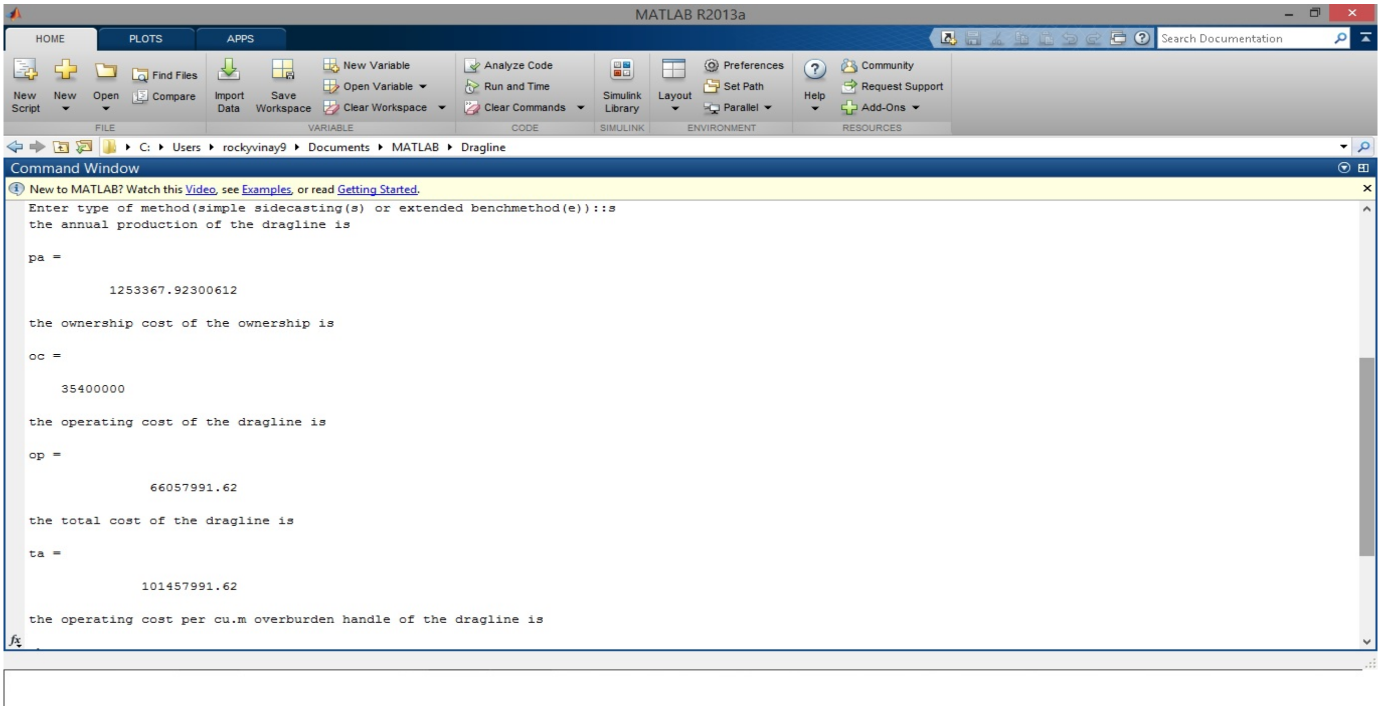


Fig 4.17: Output screenshot 14

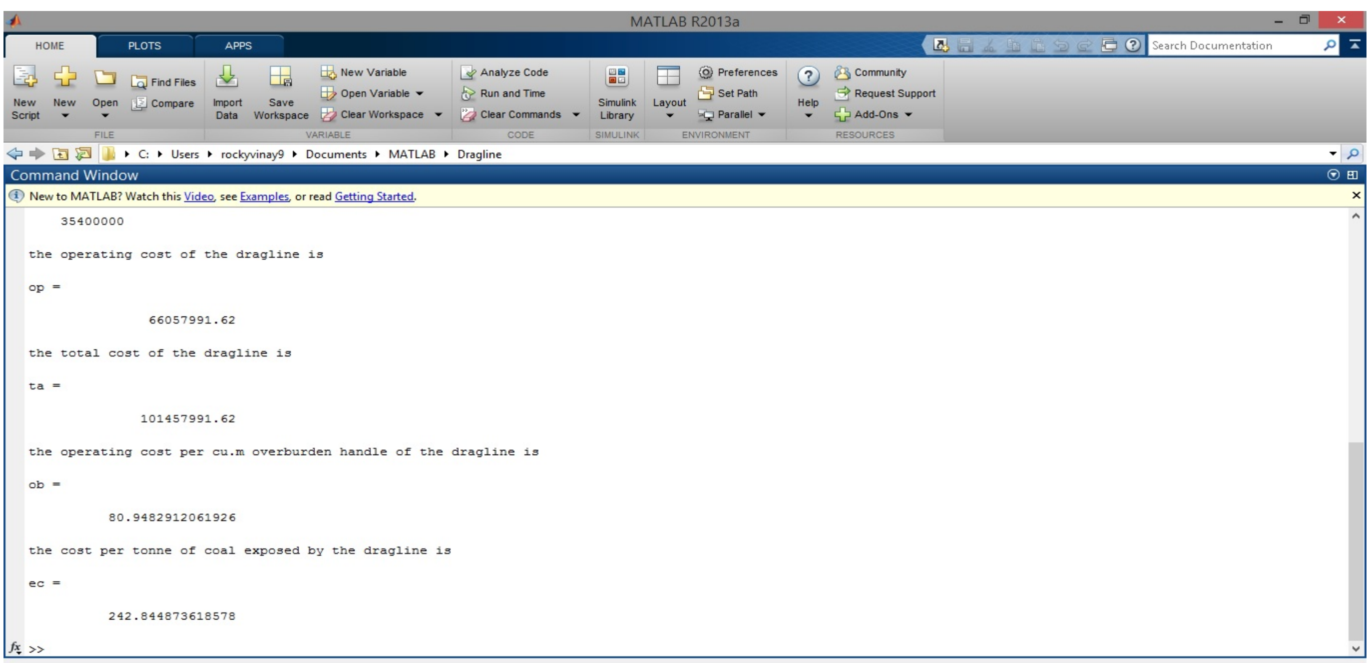


Fig 4.18: Output screenshot 15

Chapter 05

DRAGLINE PERFORMANCE STUDY

Coal India is the largest deployer of draglines in the country. However, the availability and utilization of previous year's shows that performance of draglines is decreasing.

Table 5.1- Availability & Utilization % of draglines (D/L)

	No. of D/L workings	Availability %		Utilization %	
		2012-13	2011-12	2012-13	2011-12
CIL	38/39	90	93	77	83
NCL	19	72	75	62	66
Rajnagar o/c mines (SECL)	01	84.8	86.1	68.9	68.4

Source: Annual report of CIL, NCL and SECL of year 2012-13.

Table 5.2- Analysis of age of D/L

Age in year	No. of D/L	% age
40-50	4	10
30-40	7	15
20-30	19	50
10-20	10	25
Total	40	100

5.1. ANALYSIS OF DECREASING PERFORMANCE

Productivity of draglines at NCL and other subsidiaries is largely affected by the following challenges:

5.1.1. Age of the machine (D/L): The expected life of dragline as per norms is 30 years or 1,40,000 hours. About 25% of draglines are more than 30 years old; some are even 50 years whereas 50% are in the range of 20-30 years (Refer Table 5.2). These machines have not only outlived their life but the original mines where these were planned have exhausted. As a result heavy cost is involved in maintaining these machines which are not being deployed in normal geo-mining conditions. These should be surveyed off and new machine should be deployed at suitable mines only with sufficient working area and coal reserves.

5.1.2. Maintenance: Most of the machines are imported from Russia, Ukrain or USA. Difficulty in getting the spare parts in time often lead to poor availability of draglines. Since the down time cost of dragline is very high, there should be system of "condition monitoring" of vital parts to know the impending time to replace the part/ assembly in advance before they breakdown.

Power supply to the dragline machines should be through a dedicated feeder from the substation. No other load should be given power from this feeder.

Hence good maintenance policy, which in turn depends mainly on the spare parts management, can increase the availability of draglines.

5.1.3. Utilization: Idle time of modern draglines should be minimum to get the maximum operating hours. Hot seat exchange, dedicated vehicle for shifting workers, large blast designs requiring minimal interference due to blasting, availability of dozer with operator all the time etc. are some of the measures which can be adopted for maximizing working hours.

5.1.4. Blasting: Those blasts that are difficult for the dragline to dig and require additional blasting have adverse effects on both production and cost. Poor blasting results in:

- High current in motors causing deterioration of insulation;
- High stress in drag machinery which leads to structural failures;
- Damage to the bucket;
- Breaking of drag ropes.

Blasting should be such that sufficient fragmentation is achieved. For optimum utilization of dragline, proper fragmentation of overburden is very important.

Also adopting overburden side casting through blasting by designing the blast in such a way that part of the blasted overburden is thrown directly into the de-coaled area. By doing this, substantial amount of cost can be saved.

5.1.5. Material accommodation & rehandling: Properly designed "balancing diagram" and its implementation is very important for working of draglines and its capacity utilization. Efforts should be made to reduce rehandling to a minimum extent possible. Since rehandling of material is an unproductive job, therefore, it should be kept to minimum. At the same time no coal should be left unexposed, which cannot be mined.

5.1.6. Rate of exposure of coal: Basic work of a dragline is to side cast overburden material from above the coal seam to expose coal for extraction, if the dragline bench is less, machine will be required to remove lesser material and hence coal exposure will be faster which can yield higher coal output. However, there is a tendency to give more workload to dragline, thereby resulting in slower rate of exposure and less coal production. This type of situation should be avoided.

5.1.7. Availability of skilled Manpower: Dragline mining not only requires strict discipline in the method of mining, but also requires skilled disciplined manpower operators, technicians and executives.

Operator error and poor operating techniques causes extreme ill-effects on the performance of the machine. The following poor operating techniques have to be monitored and controlled for improving the dragline operation:

- Hoist/ Drag stall
- Hoist Overload
- Hoist/ Drag slack rope
- Tight lining
- Over swing
- Fly dump
- Control lever jockeying

The operators should also be supervised how he utilizes the operational delays like shift change, wait for cable handling, dozer pushing in the roll etc. Use of simulator for operator training will increase the skill of operators.

Cost reduction will occur only when efficiency of operation is achieved. Increase in productivity of draglines can be achieved by improvements in method and pattern of excavation. Some of the factors that govern productivity are explored herein to bring out improvements in productivity.

5.1.8. Balancing diagram: Before the draglines are deploy in the mine, all the operating parameters are examined and a balancing diagram is drawn so as to get the maximum utilization of draglines. Rehandling is brought down by examining alternative methods, in reality, there exists divergence from the balancing diagram, the discrepancies between the balancing diagram and actual field performance must be thoroughly examined, planned and re-planned according to the prevailing conditions, in order to improve the productivity.

5.1.9. Digging hours: If the dragline is operated 1 extra minutes per shift that equates to 18.2 hours/year, which results in additional 10,000- 12,000 square meter of dragline production for a 24/96 dragline. Therefore, it is extremely costly to keep the dragline idle for whatsoever reason. The following are the common causes that affect the working hour, they should be minimized to get the maximum working hour:

- ✓ Shift change
- ✓ Dozing
- ✓ Relocating power cable
- ✓ Blasting
- ✓ Sitting level preparation
- ✓ Waiting on dozer for pushing the roll
- ✓ No face
- ✓ Waiting for labor, parts, tools, etc.

NCL follows hot seat exchange in all of its mines due to which time is hardly lost in shift change. Dozing operations, blasting operations, cable shifting appear to be potential areas which need proper attention for enhancing the utilization. They can be planned to carry out in maintenance periods. Idle hours can be reduced by having large blasting bank, separate power feeder, advance blasting, doing dozing work while the machine is down. All these delays should be avoided by proper resource planning. An efficient work procedure should be adopted in all the activities to maximize dragline productivity. In no case the dragline should become idle for the want of blasted material.

5.1.10. Digging pattern: Dragline has to work strictly to systematic pattern to dig out the maximum production. The dragline operator must know precisely in advance, where he is going to spot the bucket, smoothly with accuracy. During the operator change, the incoming operator has to see what previous operator was doing and simply follow on with the pattern. The operator has to

- Use layer method in spotting the bucket beginning from spoil side
- Control bucket depth, avoid stalling
- Reduce bucket fill length, avoid skimming
- Use systematic dig pattern

5.1.11. Dig rate: Dig rate is based on the maximum bucket factor, filled in shortest loading time and how well operator utilizes the hoist, drag and swing speed.

If the swing cycle is reduced by 1 second, the production increases by 90,000 cubic meters annually which is equal to 2.5% increase in the productivity. Hence the operator must use his proficiency with drag, hoist and dump functions to reduce the cycle time which contributes to the productivity of the draglines considerably.

5.2. EVALUATION OF DRAGLINE PERFORMANCE-A CASE STUDY IN RAJNAGAR OPEN CAST PROJECT

5.2.1. Introduction

Rajnagar Opencast block is located in Hasdeo Area of South Eastern Coalfields Limited, on the border of Madhya Pradesh and Chhattisgarh state and spreads over three adjoining collieries i.e. Rajnagar, West Jhagrakhand and Ramnagar Block-I, but major portion falls within the leasehold of Rajnagar R.O. Colliery. The coal deposit suitable for opencast mining is about 5.0 square km.

in area. The shape of the quarriable block is irregular due to faults. Further, siding railway track passes over the deposits which have been utilized as barriers between the quarries. Thus the faults and the railway track have divided the deposit area into 6 sectors i.e. A, B, C, D, E & F, each representing the quarry block. The nearest town Manendragarh is 8.00 kms from the project site. Mine is connected to Rajnagar by road. Manendragarh is the nearest Railway station situated on “Anuppur-Chirimiri” Branch line of S.E. Railway. The mine is situated from 23011’38.92’’ to 23013’45.03’’ Latitude & from 82005’42.16’’ to 82009’31.62’’ Longitude. The seam 13 C & 14 D is quarriable within the block. The thickness of seam 13 C varies from 3.0 m to 5.0m. The seam is generally free from dirt, bands, except one shaly coal band which ranges in thickness of 0.09 to 0.38m. The thickness of seam 14 D ranges from 2.5m to 4.0m. It consists of two bands. The upper band consists of shaly thickness from 0.11 to 1.15m. Seams 'C' & 'D' are separated by fireclay parting which ranges in thickness from 2.0m to 3.0 m. At present, mining operations are being carried out at Sector-A, E & F. 10/70 dragline is in operation in Sector - A patch & Shovel Dumper Combination in Sector E. Coal production is now being done from Sector – A dragline patch and Sector – F by contractual means (Ex-Serviceman Company). The fireclay band between the two seams is mined out separately to improve coal quality. The floor of the seam is moderately watery. However, pump of adequate capacity have been installed to take care of water accumulation, especially during the monsoon.

5.2.2. Status of Equipments

Table 5.3: Status of Equipments of Rajnagar Open Cast Project as on 01/01/2013 at Hasdeo Area

Sl. No.	**Equipment**	**Mfr Sl. No.**	**CIL Plant No.**	Date of Commissioning	Prog Hours 28/12/11	**Present Status**	**Capacity**	**Nos.**
A	**DRAGLINE**							**1**
1	DL 10/70	1/239	EXC-643	30/10/1984	175542	WORKING	10m^3	
B	**LOADING**							**4**
1	300CK POC (D1)	B-0626	EXC2527	17/11/2008	16196	WORKING	3.8m^3	
2	300CK POC (D2)	B-0653	EXC2626	29/1/2010	13974	WORKING	3.8m^3	
3	300CK POC (D3)	B-0657	EXC2627	24/2/2010	12263	WORKING	3.8m^3	

4	300CK POC (D4)	B-0660	EXC2628	31/3/2010	11981	WORKING	3.8m^3	
C	**BACKHOE**							**1**
1	TATA(HI),EX300 LCH	693	EXC2333	04/04/2005	19722	WORKING	1.57m^3	
D	**FRONT END LOADER**	TWL 3036-2589	FEL207	20/9/2010	5902	WORKING	1.9m^3	**1**
E	**DOZER**							**6**
1	D-355A-3	G-11041	T-2545	08/4/2004	26511	WORKING		
2	D-355A-3	G-11080	T-2546	28/12/2004	23850	WORKING		
3	D-355A-3	G-11588	----	17/09/2012	1703	WORKING		
5	D-155A-1	G-12409	T-2547	16/03/2005	26427	WORKING		
6	D-155A-1	G-12410	T-2548	16/03/2005	12585	WORKING		
7	D-155A-1	G-12806	T-2937	24/11/2008	33295	B/D		
F	**DRILL**							**6**
1	RECP -750E	6010101	RD-493	10/04/2001	19061	WORKING	250mm	
2	RECP C650 3	6029907	RDC-924	19/07/1999	37829	WORKING	160mm	
3	RECP C650 4	6020213	RDC-958	18/5/2003	24266	WORKING	160mm	
4	RECP 650	6020709	RDC-1094	13/08/2008	10794	WORKING	160mm	
5	IDM 30 E	2004605603	RDC-988	22/11/2004	13344	WORKING	160mm	
6	ICM 260	26008005	RDC-1124	29/07/2008	1529	WORKING	100mm	
G	**GRADER**							**2**
1	MG BG-825	3086	G-316	22/4/2004	15312	B/D	280H.P	
2	MG BG-825	3119	G-343	13/5/2006	10321	WORKING	280H.P	
H	**CRANE**							**3**
1	CRANE 75T RT880	403694	MC414	14/12/2007	2439	WORKING	75T	
2	ESCORT CRANE 8T	1532-013015	MC379	01/10/2003	9143	WORKING	8T	
3	ESCORT CRANE 3T	03/E	MC173	18/01/1989	20028	WORKING	3T	
I	**WATER TANKER**							**3**
1	W/SPRINKLER	WS28-2-255	WS128	25/07/2003	9842	WORKING	28KL	
2	W/SPRINKLER	WS28-2-264	WS152	12/04/2006	16156	B/D	28KL	
3	W/SPRINKLER(HM)	W3B00258	WS134	01/07/2005	16264	WORKING	28KL	
J	**DUMPER**							**18**
1	BH 35-2 DUMPER	499	D9601	09/06/2006	23830	WORKING	35T	
2	BH 35-2 DUMPER	500	D9602	09/06/2006	24615	B/D	35T	
3	BH 35-2 DUMPER	501	D9603	01/07/2006	18548	WORKING	35T	
4	BH 35-2 DUMPER	1051	D10946	20/07/2010	8979	WORKING	35T	
5	BH 35-2 DUMPER	1075	D10947	25/07/2010	8623	WORKING	35T	
6	BH 35-2 DUMPER	1000	D10206	06/01/2009	8775	WORKING	35T	

7	BH 35-2 DUMPER	1020	D10208	31/8/2009	10388	WORKING	35T	
8	HM 1035 DUMPER	B3B00322	D9189	26/04/2005	14547	B/D	35T	
9	HM 1035 DUMPER	B3B00521	D9599	04/08/2006	16979	WORKING	35T	
10	HM 1035 DUMPER	B3B00522	D9600	04/08/2006	16209	WORKING	35T	
11	HM 1035 DUMPER	B3B00584	D9975	28/03/2007	15376	WORKING	35T	
12	HM 1035 DUMPER	B3B00585	D9976	28/03/2007	21810	WORKING	35T	
13	HM 1035 DUMPER	B3B00586	D9977	28/03/2007	15869	WORKING	35T	
14	HM 1035 DUMPER	B3B00678	D10370	01/02/2009	15317	WORKING	35T	
15	HM 1035 DUMPER	B3B00679	D10371	15/01/2009	10426	WORKING	35T	
16	HM 1035 DUMPER	B3B00680	D10372	15/01/2009	16203	WORKING	35T	
17	HM 1035 DUMPER	B3B00681	D10373	15/01/2009	16932	WORKING	35T	
18	HM 1035 DUMPER	B3B00683	D10374	27/01/2009	14228	WORKING	35T	

5.2.3. GENERAL CONDITIONS/ INFORMATION OF THE MINES

1. Name of the Mine : **RAJNAGAR OPENCAST**
2. Name of Units : SECTOR 'E' & SECTOR 'F'
3. Date of Opening : SECTOR 'E' – 01/01/1998
   SECTOR 'F' -- 23/01/2007
4. Location of the Mine : The mine is situated from $23^0$11'38.92'' to $23^0$13'45.03'' Latitude & from $82^0$05'42.16'' to $82^0$09'31.62'' Longitude
5. Total area of leasehold : 842.21 Hectares

6. Name of Seam with the total coal reserve

Table 5.4: Name of Seam with the total coal reserve

Seams	**Thickness (In Mtr.)**	**Reserve (In Million Tones)**	**Total**
14D	2.50 to 3.50	2.00	5.00
13 C	3.00 to 5.00	3.00	

7. Geological & mining conditions

Table 5.5: Number and Name of the Seam being worked (with stages of working)

Sl. No.	**No. & Name of the Seam**	**Thickness (in mtr)**	**Depth of Seam**	
			Sector - E	**Sector - F**
1	14D seam	2.50 to 3.50	40 Mtrs.	70 Mtrs.
2	13C seam	3.00 to 5.00	45 Mtrs.	75 Mtrs.

8. Details of Outlets

Table 5.6: Details of Outlets

Name / Purpose	Area of Cross Section	Length / Depth		Working of different seams connected with
		Sector-'E'	Length – 1200m Depth – 75m	13C & 14D
		Sector-'F'	Length – 1000m Depth -- 50m	13C & 14D

9. Geological Disturbances

1.1.1. Dyke … … … NIL

1.1.2. Faults … … … 14 Nos.

5.2.4. Selection of Dragline

5.2.4.1 Conditions for operation of the dragline

a. Gradients flatter than 1 in 6

b. Seams should be free of faults & other geological disturbances

c. Deposits with major strike length

d. Thick seams with more than 25m thick are not suitable

e. A hilly property is not suitable

Dragline removes over burden to gain access to coal seam. Draglines do not mine ore. The coal is then removed from the mine using stripping shovels and dump trucks. Types of draglines are:

1. Crawler mounted draglines
2. Wheel Mounted draglines
3. Walking draglines

Size of dragline is indicated by the size of bucket expressed in cubic meter. Most of the drag lines handle different size of bucket depending on the boom and class of material excavated. Maximum lifting capacity of the dragline is limited to force which will lift the machine over. Size of bucket is to be reduced when ling boom is used or when material has a high specific gravity. In practice the combined weight of the bucket and its load should produce tilting or overturning forces not greater than 75% of force required to tilt the machine boom if the material is difficult to excavate, the use of smaller bucket which will reduce digging resistance.

5.2.4.2 Russian Dragline

Boom is main structural built up columns subjected to forces in direction of booms and buckling in direction of booms and buckling in perpendicular direction. All the forces are transferred through pulleys. It shows different position of buckets for digging operation and poor digging takes 42% more power and force on pulley at the end transferred to compression members.

5.2.4.3 Drag Cable, Hoist Cable, Dump Cable (ropes)

This is essentially a tension member as rope as per specifications made of high strength wires of 6-7 cores or locked coil winding rope. Following factors are to be considered for its durability. Tension ropes are very efficient, flexible, dependable members. Hence dragline consists of structural frames as compression members taking buckling into considerable and rope as efficient tension member. Mining rope built to withstand sever conditions is made of quality steel. The wires are drawn from acid open hearth steel with C= 0.5 to 0.8 Mh 0 .3 to 0.7 Si 0.2% S & P should not exceed 0.04% (BS 236 or IS 1835). Wire tensile strength will normally be 160 grades (160 kg/mm^2 to 189 kg/mm^2). Wire tensile strength is 160 to 180 kg/mm^2. Young's modulus of wires in tension = 1.65 to 2x10^6 kg/sq. cm. Wire diameter 0.2 to 0.1mm toughness of wire obtained by tempering and tensile strength is increased by repeated drawings.

5.2.4.4 Rope Lay

Rope lay is usually designated numerically by stating number of strands and number of wires per strands. The 6 strands of 7 wires per strand for 6 X 7 rope. The strands are laid around a core or a wire strand core (WSC) or an independent wire rope (IWRC). The wires forming the strand and strands forming the rope are twisted on the opposite direction in an ordinary lay rope. Then both wires and the strands are twisted in same directions it is called Langs lay rope. In LOCKED COIL CONSTRUCTION each layer of wires are spun in a helix about the Centre core. Outer layers are laid in opposite direction of the inner layer, with the result that the rope is non-spinning. Factors of safety are usually 7 to 10 for winding ropes. According to Indian Mining Regulation (MCR 80 (d)) it is 10 capacities. Factor F_o = Breaking load of rope and the total load suspended.

5.2.4.5 Cable or Rope Selection

The knowledge of rope and properties are to be known from manufacture's table.

$$\text{Rope Dia.} = \sqrt{\frac{\text{WFL}}{\text{K(lo}-\text{FH)}}} \text{ or } d = \sqrt{\frac{\text{WFL}}{\text{K}-\text{CHF}}}$$

Where,

W = End load static and dynamic force on cable or tensile force including impact

F = Factor of safety, L_o = Snap length, $K_{0\,=}$ Rope constant

H = Lifting depth for hoist and drag cable in m.

Fixed cable in tensile attached boom are larger in diameters while movable swinging tensile cables are small diameter number of cable. More number of small diameter cables is preferred in view of high efficiency.

5.2.4.6 Hoisting and Drag Chain

Chains are the main draggers of bucket providing for dragging by the bucket. These chains shall conform to IS 7587 (Part 1 to Part 8 (1975-1984) and IS 10131-1992. U link is closed circuit, C hooks. Shackle type chains are used. Mode of high tensile steel for coal ploughs in mine and for dragging. The buckets chains are galvanized chains are flexible and hence exclusively used only for dragging and not for lifting of materials in draglines.

5.2.4.7 Buckets

In selecting the size and type of bucket the dragline and bucket should match. Buckets are usually available in three sizes

- Light
- Medium
- Heavy duty

The light buckets are used for excavation in sandy, clay or sand, medium buckets are for soft shale loose gravel and heavy duty buckets are used for minimum stripping and highly abrasive materials. In selecting the most suitable size of bucket for use with given dragline, properties of buckets like volume, weight are considered. Larger size of bucket is recommended. Weight of bucket and material handled decides the safe load capacity recommended for dragline.

5.2.5. Principals of Load Combinations for Draglines Structure

Principals of load combinations and applicable factor of safety for steel boom, wire dragline or pipe rope draglines for mines under normal and special operating conditions have been formulated by IS (IS 13834, IS 3177, and IS 8686 for cranes) IS 875 Part 1 to 4, 1994.

5.2.5.1 Dead Loads

Dead loads consist of all weights and components like boom, rope chain bucket and structure attached to the boom. The weight of material and the heaviest material lifted.

5.2.5.2 Static Loads

This consists of pay load weight of counter weights, lock coil rope all dead loads.

5.2.5.3 Winding Loads

All pay loads of material lifted including weight of bucket.

5.2.5.4 Drag Forces

This is most important depends on type of material dragged. Harder the material and stiff is the angle of dragging greater is the force. This force is cutting force of bucket resulting into tension in chain and drag cable. This force is transferred to fair lead in dragline transferred inclined force at hinged end of boom

5.2.5.5 Dynamic Loads

For loads of accelerations, swing in 360^o, jerks, impact on the structure, dynamic load is very difficult to determine. Effect of acceleration on rope is much difficult and much higher. However an empirical formula fairly gives a dynamic load.

$$T_d = \frac{1.72\, a\, (W+A)}{g}$$

Where, a = acceleration/retardation, g = acceleration due to gravity 9.81 m/sec, w = static load + pay loads + weight of bucket, A = weight of rope, T_d= dynamic force

5.2.5.6 Breaking load of Rope and Impact

In dragline, more number of smaller dia. ropes is used and hence breaking load may be taken for minimum number of wires. Impact of 33% to be added in breaking loads. Manufacturer's usually supplied the data of breaking strength. This force is transferred to one of the hinge support resulting into horizontal and vertical loads and forces in boom.

Frictional load: - This is assumed to be 1% of the static and dynamic load on dragline.

5.2.5.7 Wind Loads

Wind load shall be taken as per IS 875 part III with appropriate location and terrene on the boom of dragline

$$P_z = 0.6\, V_z^2$$

Where, P_z = Pressure N/mm^2, V_z= Velocity m/sec.

5.2.6. Specification of HKM3

Make – USSR, Model- 10/70, Walking Dragline, and Bucket Capacity- 10m^3, Boom Length- 70m, Walking Speed-0.2 km/hr (200mtr/hr), Maximum 10^o gradients can walk, Machine wt- 756te, Maximum digging depth- 35.0mtr, and Maximum dumping height- 27.5mm, Maximum Dumping Radius- 66.5m, Width of the M/C – 13.2, Capacity – 1.12 million m^3 O.B. per year. Operation cycle time – 54 second (for max. 135 degree swing), Average buckets/Hr. – 66, Power consumption per minute – Rs 40/-, Lighting arrangements by 110 Volt D.C. supply.

5.2.6.1 Rope Dimensions

Table 5.7: Rope Dimensions

Type of Rope	**Diameter(In mm)**	**Length (In mtr)**	**No. of Ropes**
Hoist Rope	38	165	2
Drag Rope	52	95	2
Boom Suspension Rope	52	106	2
Dump Rope	38	9.5	1
Boom Rope for Up & Down Movement	24	120	1
Jeep Crane Rope	13	100	1

5.2.6.2 Motor Capacity

Table 5.8: Motor Capacity

Type of Motor	**No. of motor**	**Capacity (In KW)**	**Power Supply**
Main Motor	1	1250	AC 6 K.V.
Swing Motor	2	400	DC 440 V
Hoist Motor	2	450	DC 440 V
Drive Motor	2	450	DC 440 V
Walking Motor	1	450	DC 440 V
DC Generator	3	900@1000 rpm	DC 440 V

5.2.7. Performance of 10/70 Dragline

The machine is intended for excavation of soil (OB) in stripping operation without the use of transport vehicles, with dumping of the soil into the workout area or on the pit side. The 10/70 excavator dragline is a full swing electrically driven earth moving machine with a walking travel mechanism and a dragline in front end attachment.

5.2.7.1 Month wise production comparison of two years

Table 5.9: Month wise production considering month from 26th to 25th

	Production in m^3	
Month	2012-13	2011-12
April	37800	147900
May	77700	172200
June	91800	150600
July	90300	152600
Aug	72000	165900
Sept	93300	112200
Oct	97800	213500
Nov	60600	108000
Dec	108000	21600
Jan	130500	111960
Feb	108900	91800
March	224540	35040
Total	1193240	1483300

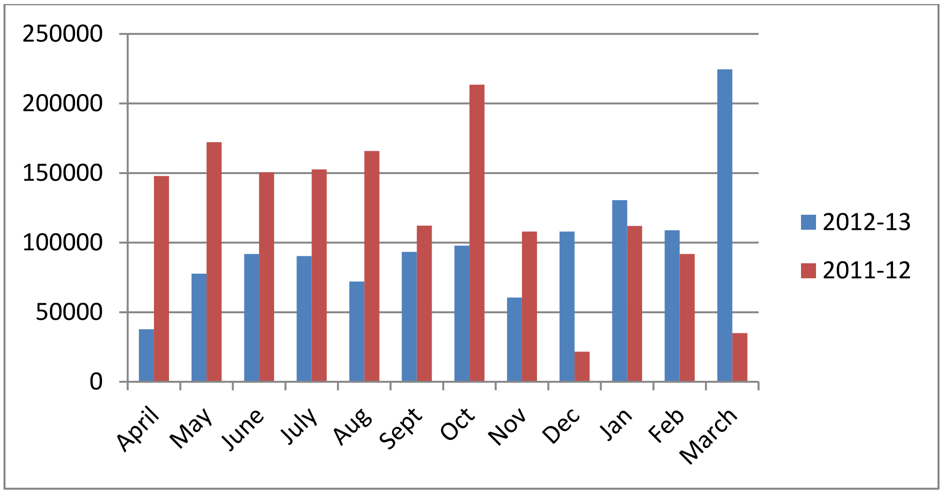


Figure 5.1: Month wise production comparison of two years

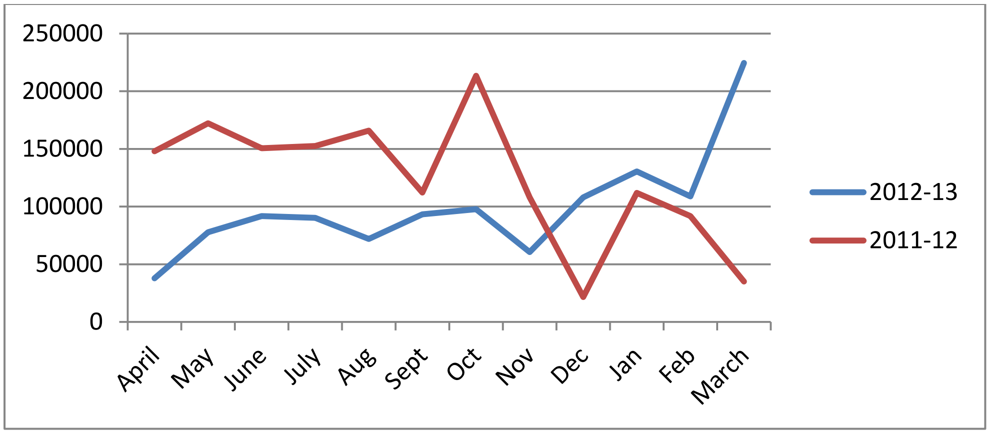


Figure 5.2: Graphical representation of production data

5.2.7.2 Evaluation of Availability and Utilization

As per the CMPDI norms the maximum availability of the M/C is 84% and the utilization is 80% with a minimum output of 1.12 million cubic meters OB per year. To evaluate the Availability (A) and Utilization (U) the field data acquired was substituted in the Equations (i) and (ii).

$$A = \frac{SSH - (MH+BH)}{SSH} \times 100 \quad \ldots\ldots \text{(i)}$$

$$U = \frac{SSH - (MH+BH+IH)}{SSH} \times 100 \quad \ldots\ldots \text{(ii)}$$

Where, SSH is scheduled shift hours, MH is maintenance hours, BH is breakdown hours, IH is idle hours

5.2.7.3 Month wise working Hrs. (Considering month from 26th to 25th)

Table 5.10: Month wise working Hrs. (Considering month from 26th to 25th)

2012-13								
Month	**S.S.H.**	**W.H.**	**M.H.**	**B.D.**	**I.H.**	**Total**	**Availability**	**Utilisation**
April	744	229	40	0	475	744	94.6	30.8
May	720	511	58	26	125	720	88.3	71.0
June	744	507	59	27	151	744	88.4	68.1
July	720	432	38	209	41	720	65.7	60.0
Aug	744	483	56	40	165	744	87.1	64.9
Sept	744	534	59	78	73	744	81.6	71.8
Oct	720	551	51	18	100	720	90.4	76.5

Nov	744	364	196	122	62	744	57.3	48.9
Dec	720	601	57	33	29	720	87.5	83.5
Jan	744	582	56	4	102	744	91.9	78.2
Feb	744	618	60	6	60	744	91.1	83.1
March	672	620	40	0	12	672	94.0	92.3
Total	8760	6032	770	563	1395	8760	84.8	68.9

2011-12								
Month	**S.S.H.**	**W.H.**	**M.H.**	**B.D.**	**I.H.**	**Total**	**Availability**	**Utilisation**
April	744	555	27	82	80	744	85.3	74.6
May	720	597	36	26	61	720	91.4	82.9
June	744	537	28	95	84	744	83.5	72.2
July	720	549	44	85	42	720	82.1	76.3
Aug	744	604	63	0	77	744	91.5	81.2
Sept	744	559	62	16	107	744	89.5	75.1
Oct	720	522	88	0	110	720	87.8	72.5
Nov	744	531	90	4	119	744	87.4	71.4
Dec	720	186	53	4	477	720	92.1	25.8
Jan	744	484	57	12	191	744	90.7	65.1
Feb	744	579	52	4	103	738	92.5	77.8
March	696	305	258	32	71	666	58.3	43.8
Total	8784	6008	858	360	1522	8748	86.1	68.4

5.2.7.4 Comparison of W.H. of 2012-13 & 2011-2012

Table 5.11: Comparison of W.H. of 2012-13 & 2011-2012

	2012-2013	**2011-2012**
Month	W.H.	W.H.
April	229	555
May	511	597
June	507	537
July	432	549

Aug	483	604
Sept	534	559
Oct	551	522
Nov	364	531
Dec	601	186
Jan	582	484
Feb	618	579
March	620	305
Total	6032	6008

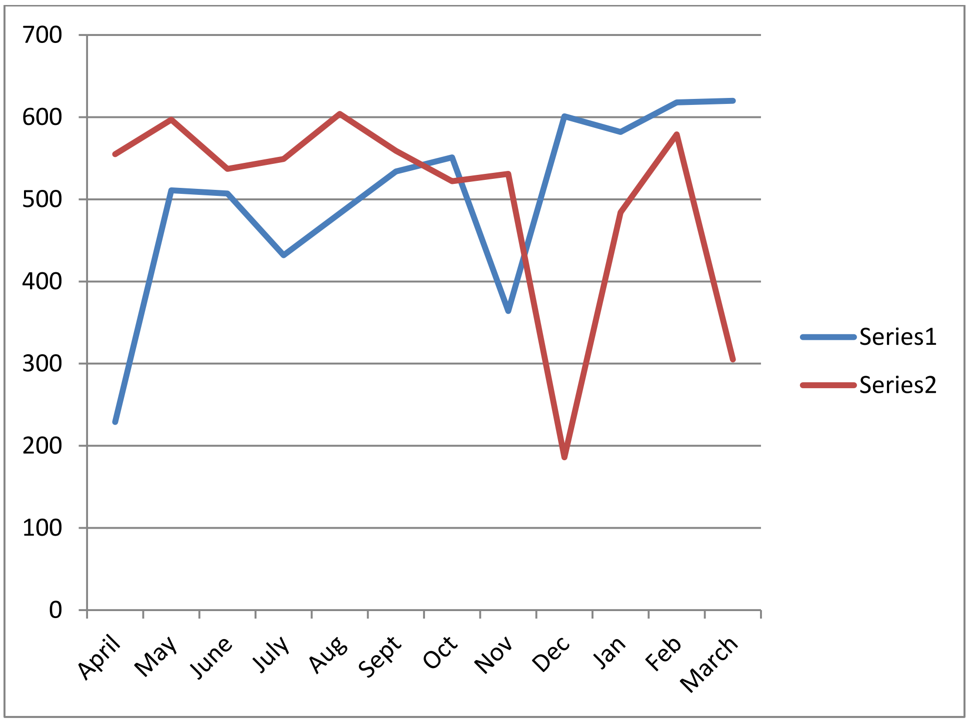


Figure 5.3: Graphical comparison of W.H. of 2012-13 & 2011-2012

5.2.8. Stability and Safety of Dragline Structure

Due to sustained loading operational loading steel structure boom and rope develops distress. The reasons of distress are:

1. Configuration change
2. Incorrect detailing of steel
3. Changes in configuration of structure

4. Incorrect fabrication
5. Corrosion of steel
6. Changes in physical and geometrical properties of structure and rope
7. Removal of member of joints
8. Un-presented loads cyclone, earthquake, bomb attack, etc.
9. Many more environmental factors
10. Change in support conditions of sheave, rope, forces, pulleys

Above factors change the stable nature of structure under static and dynamic loads. Stability is defined as physical non movement of structure and resultant zero of all the forces. However minimum movement for structure is prescribed which do not alter the stability of structure.

5.2.8.1 Factors effecting stability

a. Excessive deflection of frame in any direction.
b. Physical rotation of structure.
c. Functional use is altered and the geometry of boom configuration is changed.
d. Buckling of individual members and displacement of joints for boom structure.
e. Opening of welds bolts seepage.
f. Cracks in welds.
g. Un-presented loads hung to individual member.
h. Corrosion and change of E, m and properties of steel section, pipes rope.
i. Integrity and efficiency loses or clamps joints, welds
j. Reliability is lost
k. Dynamic movement (physical)

5.2.8.2 Structural Safety Audit for Dragline Structure

Durability and serviceability of structure depends on constant watch on structure safe and sound structure is to satisfy the strength, stability, durability and serviceability criteria discussed earlier in details. Material properties are changed during service life. Following important factors area to be tested from times to time and verified with the original safe design at the time of construction.

5.2.8.3 Stability Checks

1. Deflection, rotation and physical movement in all direction should be measured by applying loads.
2. Deformation of frame, buckling of each individual members cracks or defects in joint are to be checked.
3. Plastic stage of selections used should be checked by observing change in color and ductility of section.
4. Stress, strains, young's modulus, poison's ratio should be calculated on samples by destructive method taking care that the structure is not affected by removing any member.
5. External redundancy, internal redundancy and equilibrium conditions of loads and reaction must be checked.
6. Temperature distortion, creep distortion, of individual member should be checked.

5.2.8.4 Durability Checks

1. Corrosion of steel and M.S pipe is to be measured finding out the damage done to members and change of geometrical properties including buckling.
2. Acid rains in mines are to be studied and damage of steel is to be assessed.
3. Coating applied is checked for diffusibility and thickness calculated.
4. Cracks, pitting of steel to be assessed for service life.
5. Plastic hinge formation checked excessive strain.
6. Young's modulus, Poison's ratio, stiffness of structural member, pipes and ropes should be ascertained.

5.2.8.5 Serviceability Checks

1. The structure should respond for the purpose for which it is designed.
2. Structure must be usable in working order.
3. Swings sways, guide rails, counter weight should be checked which transfer load to structure in perfect alignment.
4. Minimum or no maintenance condition.
5. Efficient response of frame and member subject to static or dynamic loads.
6. Clean, elegant looking, noiseless during movement.

7. Increased efficiency of operation. It is recommended that structural safety audit must be done every five years for long life stability and efficient structure. This audit increases the reliability of structure.

5.2.8.6 SAFE OPERATION OF DRAGLINE

For a safe operation of draglines which consists of walking and digging, the following conditions must be adhered to:

A-WHILE WALKING

- There are no rocks on the formation to damage the base, walk shoes or cut the trailing cable.
- Walking must not be undertaken on an incline of more than 1 in 10.
- Ground conditions are suitable and fully compacted with level difference between two shoes is ±100 mm.
- There is sufficient room to rotate the dragline to enable it to walk.
- During level walking, where the bucket is carried at about one third radius, while walking up a gradient the bucket is brought close to the fair leads with rear of the bucket just resting on the ground and drag brake applied.
- In case of very soft ground, where in spite of compacting, the soil is spongy-the bucket is parked on the ground with the drag ropes left slack enough to allow several steps to be taken before movement of the bucket becomes necessary. In case, while walking on soft grounds there is build-up of material under the base, this is to be removed one or two steps sideways.
- Keeping all controls in neutral position, hoist, drag, swing brakes have been applied.
- There is a full crew with sufficient number to handle the trailing cable; one responsible person is in-charge on the ground to give directions to the operator and crew. During walking, it is to be ensured that there is sufficient slack in the trailing cable, and it is kept clear of base and walk shoes. In addition, there are no kinks in the trailing cable.
- Before walking, the position of the dragline- where it is required to be placed must be as curtained and accordingly in the direction of travel, the rear of the machine is pointed.

Where as in case of walking mechanism being governed by hydraulic system, proper functioning of main and auxiliary cylinder, opening and closing of locks on which shoes rest, proper functioning of control valves, pumps and limit switches are to be ensured, in case of mechanical walking system, it is to be ensured that power from drag to walk has been transferred and all the connecting links (in case of mechanical transfer of power i.e. same gear-box for both drag and walk) are properly functioning.

B-WHILE WORKING

- Ensure no water passes beneath dragline, failing which there is possibility of sliding of the machine;
- Before dumping O/B in de-coaled area, the de-coaled area to be de-watered, failing which the entire O/B mass dumped may slide when heavy dragline is placed above it;
- Ensure that no undercut is made below the machine;
- Dumping height should not be too high, to avoid strain on machine;
- Swing of the machine should be designed in such a way that it should be within 100 degrees;
- Length of drag rope to be optimum to avoid unnecessary dragging of bucket, to not only to reduce wear on bucket but also to minimize cycle time;
- Floor where machine is proposed to place should not be hard (un-blasted), this allows the machine skid and also while walking there will be load on marching shoe. However the floor should be made firm by moving dozer over the floor;
- Machine should be placed on level ground to avoid wear on swing shafts as well as swing pinions;
- Avoid hard digging, can be achieved by good blasts;
- Regular maintenance of machine improves availability and life.

Chapter 06

RESULTS

- The availability of the Singareni OCP-I dragline is 78.88%
- The availability of the Samaleswari dragline is 83.33%
- The availability of the Rajnagar OC dragline is 84.8%
- The utilization of the Singareni OCP-I dragline is 70.41%
- The utilization of the Samaleswari dragline is 75%
- The utilization of the Rajnagar OC dragline is 68.9%
- The projected annual output of the Singareni OCP-I dragline is 2.807 M cu.m.
- The projected annual output of the Samaleswari dragline is 1.253 M cu.m.
- Dragline operating cost per m3 overburden handle considering annual output of Singareni OC-I dragline (24/96) as 2.807 M cu.m is Rs. 81.67
- Dragline operating cost per m3 overburden handle considering annual output of Samaleswari dragline (10/70) as 1.253 M cu.m is Rs. 82.2
- Estimated cost/tonne of coal exposed by the Singareni OC-I dragline(24/96) is Rs. 285.78
- Estimated cost/tonne of coal exposed by the Samaleswari dragline(10/70) is Rs. 247.17.

Chapter 07

CONCLUSION

Draglines are very efficient, high productive, low cost excavating machines. We should try to get maximum output from these machines by adopting safe work practices. No short cuts should be applied, otherwise it may prove dangerous. The trained manpower should be rotated among the similar mines, in order to gain fully utilize their training and expertise. Dragline structures are generally not maintained for serviceability. Some of the members are removed for the convenience and mining operation making structure unstable. Initial defects of fabrication of bolts, rivets are enlarged. Improper coating for protection of structure leads to corrosion of steel and reduction of properties of section environmental effects, mishandling of structure leads to deterioration of structures. Following conclusions are drawn from this study:

Factors affecting the production and cost of coal exposed by dragline are:-

- ✓ Increased no. of idle hours due to non-availability of blasted muck pile, operator availability, ability and performance
- ✓ Increased breakdown time
- ✓ Increased breakdown and maintenance costs
- ✓ Improper supply of spare parts of dragline

Scope for improvement

- ✓ Conditions assessment must be done in every five year.
- ✓ Retrofitting and replacing member and rope must be done in every five year.
- ✓ Damage and stability assessment should be done in every five year.
- ✓ Increasing the Dragline Productivity through Maximizing Cast by using blasting techniques like i-kon system of blasting.
- ✓ Employing a spare dragline operator and reducing the Variability in dragline operator performance.
- ✓ Better preventive maintenance schedule to reduce the breakdown time and breakdown costs.

- ✓ Strengthening of structure must be done with modern methods.
- ✓ Coatings and protective coats must be applied to structures.
- ✓ Instability must be removed by balancing the structures for loads and configuration.
- ✓ All obstructions to be remove smooth functioning and serviceability.
- ✓ Structure must look elegant, smart and fully functional.

Chapter 08

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